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STORAGE AND DISPOSAL OF IRON ORE PROCESSING WASTEWATER

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16. *Abstract* This study was concerned with the handling of wastewater resulting from the concentration of low grade iron ore. Specific objectives were to develop tailings basin design and management criteria, to investigate methods of treating basin effluents, and to investigate transport characteristics of particulate materials contained in basin discharges. These objectives were accomplished through laboratory and field studies conducted at the tailings impoundment systems associated with two iron ore concentrating plants located in Upper Michigan. Annual water balances were formulated for the tailings system at each plant to show the relative importance of precipitation, surface outflow and seepage. Settling column and dye dispersion tests were employed to predict concentrations of suspended material remaining in basin effluents. The cost effectiveness of various coagulation systems for treating basin effluent was investigated. Particular attention was devoted to a study of the thickening and dewatering characteristics of the slurry produced by chemical coagulation of the basin effluent. Synthesis of alternative systems for handling tailings wastewater was hampered by a lack of information on water quality requirements for reuse within ore concentrating processes. Finally, settling column experiments in which fine tailings particles were diluted with various natural waters were conducted. (EPA)

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March 1974

STORAGE AND DISPOSAL
OF
IRON ORE PROCESSING WASTEWATER

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ABSTRACT

The overall objective of this investigation was to advance and improve the applied technology related to the storage and disposal of wastewater resulting from the concentration of low grade iron ore. The investigation was based both upon laboratory studies and upon field studies conducted at the tailings impoundment and clarification systems associated with two iron ore concentrating plants located at the Republic and Empire Mines in the Upper Peninsula of Michigan.

Annual water balances were formulated for the tailings impoundment systems associated with each plant. Particular attention was directed toward estimating design flows and associated seasonal storages required during the critical spring melt period. Dye dispersion tests conducted on three clarification basins indicated considerable stagnant volume and short circuiting. For flocculent suspensions, settling column analyses were used in conjunction with the dye dispersion curve to predict effluent solids concentrations with a correlation coefficient of 0.882. Application of basin volume correction factors ranging from 2.5 to 10. was suggested when the settling column analysis was applied by assuming a plug flow hydraulic regime.

A series of jar tests was conducted using the Republic Mine tailings basin effluent to investigate the cost effectiveness of alum and several polymeric coagulants. Under most conditions, the cationic polymer coagulant, Calgon M-510 was found to be the most economical, with chemical costs ranging from $2.50/MG to $8.00/MG depending on initial turbidity and temperature. However, other coagulants tended to produce less sludge volume. An investigation of several sludge thickening aids showed the anionic polymer, Calgon M-560, to be the most effective in reducing the unit area required for thickening. Cationic polymer, Calgon M-510, was an effective vacuum filtration aid when applied to a sludge prethickened with Calgon M-560.

A possible system to collect, thicken, filter and dry the 16.9 tons/day of coagulated solids produced at the Republic Mine was synthesized. The initial investment was estimated as $870,800, and the cost per ton of solids was estimated as $37.65. The synthesis of other alternative systems for dealing with these solids was hampered by the lack of information on water quality requirements for reuse within concentrating processes.

Settling column experiments in which the fine tailings were diluted with various natural waters were conducted to determine the influence of temperature, dilution, dissolved solids and alum residual. These factors were found to have greater effects in the case of fine particles (average size of 1.3 microns) resulting from a hematite ore than in the case of larger particles (average size of 6.5 to 8. microns) resulting from a magnetite ore.

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SECTION I
CONCLUSIONS

1. For tailings impoundment and clarification systems located in the Lake Superior iron mining district, the critical flows for hydraulic and clarification design occur during the spring snow melt period.

2. Annual water balances on two operating impoundment systems indicated that:
   a) The volume of excess flow released during the spring melt period resulted from snow accumulated both on the impoundment and on the tributary natural watershed and from tailings ice and possible ground water storage.
   b) Based on unit areas of the impoundment system, the accumulation of tailings ice and possible ground water storage ranged from 10.1 inches to 13.7 inches of water equivalent. The higher value was associated with a system in which overland sheet flow occurred over a greater area.
   c) Based on unit areas of the impoundment system the average seepage rate varied from 60 inches/year for a system with a total hydraulic loading of 552 inches/year to 93 inches/year for a system with a total hydraulic loading of 159 inches/year. At the lower hydraulic loading, seepage assumed greater significance in the water balance.

3. In the design of a tailings clarification system, the design outflow rate is related to the seasonal storage provided. Both the design flow for clarification and the associated seasonal storage can be estimated by a straightforward hydrologic analysis of the system. This analysis involves the synthesis of an approximate spring hydrograph associated with some recurrence interval. Data required for this analysis include records of precipitation, snowpack, and runoff applicable to the natural watershed tributary to the tailings system. In addition allowances must be made for tailings ice and seepage. At present, there are few data which can be used to formulate estimates of tailings ice and seepage.

4. The results of dye dispersion tests on three clarification basins indicated considerable stagnant volume and short circuiting as the ratio of the time to the peak dye concentration divided by the theoretical detention time ranged from 0.071 to 0.29. The degree of short circuiting was qualitatively related to the plan configuration of the basins.

5. The Republic Mine tailings waste was generated from a hematite ore by a flotation process. This waste with an initial solids content of about 8% and no added coagulant settled as a flocculent suspension and exhibited a color similar to tomato juice. For this waste, settling column analyses could

be used in conjunction with the basin residence time distribution to predict the effluent suspended solids concentration with reasonable accuracy. The correlation coefficient between observed and predicted values was 0.882. For design purposes, in the absence of information on the expected residence time distribution, the quiescent settling column analysis could be employed directly. However, the theoretical basin detention time determined in this manner (for a given basin depth) must be multiplied by a correction factor. A correction factor of approximately 2.5 was determined for critical conditions at the Republic Mine clarification basin. The residence time distributions of the other basins studied suggested correction factors ranging from 6 to 10.

6. The Empire Mine tailings waste was generated from a magnetite ore by a magnetic separation process. This waste, with an initial solids content of 46% and a residual concentration of thickening aid, quickly developed an interface during settling tests and subsequently settled as a hindered suspension. The supernate above the interface contained less than 5mg/l of suspended solids whereas the effluent from the acutal pond, operated at an overflow rate of approximately 1% of the interface subsidence rate, ranged from 10 mg/l to 20 mg/l. In this case, the suspended solids concentration remaining within the supernate could be used to approximate the concentration in the pond effluent within 15 to 20 mg/l.

7. A series of jar test was conducted using the Republic Mine tailings basin effluent to investigate the cost-effectiveness of alum and several polymeric coagulants at an initial turbidity of 1400 JTU and temperature of 13° Centigrade. Alum, certain cationic polymer coagulants (Calgon M-500, M-510 and M-520) and certain cationic, anionic and non-ionic coagulant aids (Nalco Na-603, Dow A-22, and Dow N-11) used in conjunction with alum were found to be effective in reducing the initial turbidity by 90% or more. Anionic and non-ionic polymer coagulants were less effective. Of the coagulants or coagulant combinations which reduced the initial turbidity by 90% or more, the most effective cationic polymer coagulant was Calgon M-510 and the most effective coagulant aid was Dow non-ionic polymer N-11.

8. A series of jar tests was conducted using the Republic Mine tailings basin effluent to investigate the cost effectiveness of alum, cationic polymer coagulant Calgon M-510 and non-ionic polymer coagulant aid Dow N-11 plus alum at various levels of initial turbidity and termperature (400 to 1400 JTU and 1°C to 25°C). No single coagulant or combination of coagulants was found to be most economical in achieving a given supernatant turbidity under all conditions of initial turbidity and temperature. Chemcial costs to achieve a specified turbidity varied greatly with initial turbidity and temperature. However, for supernatant turbidities greater

than approximately 60 JTU, Calgon M-510 was the most economical under all conditions of initial turbidity and temperature. Approximate chemical costs for Calgon M-510 required to attain a supernatant turbidity of 60 JTU after 30 minutes settling time ranged from $2.50 per million gallons ($3.00/ton dry solids, dosage of 0.68 ppm) at an initial turbidity of 400 JTU and temperature of 25°C to $8.00 per million gallons ($3.10/ton dry solids, dosage of 2.18 ppm) at an initial turbidity of 1400 JTU and temperature of 1°C. At low initial turbidities (400 JTU) this cost, to achieve a supernatant turbidity of 60 JTU, was virtually equal to the cost of alum plus coagulant aid Dow N-11 and was about 80% of the cost of alum alone. However, at high initial turbidities (1400 JTU), the cost of Calgon M-510 to achieve a supernatant turbidity of 60 JTU was roughly 50% of the cost of alum plus coagulant aid Dow N-11 and roughly 35% of the cost of alum alone.

9. Differences were observed between the volumes of sludge generated during jar tests employing various coagulants. At an initial turbidity of 1400 JTU (700 mg/l suspended solids) the cationic coagulant aid, Nalco Na-603 employed in conjunction with alum (3.1 ppm Na-603, 50 ppm alum, chemical cost of $18.70/MG, supernatant turbidity of 47 JTU) generated about 10.5 ml sludge/liter. This amounted to about 45% of the sludge volume generated by the most economical coagulant, Calgon M-510 (3 ppm, chemical cost of $11.00/MG, supernatant turbidity of 28 JTU) under identical conditions. Volumes of sludge generated by other coagulants generally fell between the volumes produced by Calgon M-510 and Nalco Na 603.

10. An analysis of the coagulation - flocculation system at the Republic Mine suggested that improvements could be made to bring the actual values of the mean velocity gradient and detention time closer to their optimal values. Since the hydraulics of the present system are largely set by the natural terrain, a significant improvement would probably require the construction of a mechanical rapid mix - flocculation system.

11. At the Republic Mine, a large volume (1.44×10^6 cubic feet) of sludge with an average solids content of 22.6% had accumulated in an impoundment as a result of chemical coagulation of the highly colored red water leaving the tailings basin. During the 1 year study period these solids accumulated at an average rate of 16.9 tons of dry solids per day. An investigation of the thickening and filtration characteristics of this sludge was conducted to assess the feasibility of thickening and dewatering this and any future solids accumulation.

a) An investigation of several thickening aids showed the anionic polymer Calgon M-560 at a concentration of 0.05% (by weight of solids) to be the most effective in reducing the unit area requirement. A unit area of 2080 ft^2 sec/lb was estimated for a feed solids concentration of 10% and an underflow concentration of 20% in the presence of 0.05% Calgon M-560. This amounted to 23% of the unit area estimated for identical conditions in the absence of thickening aid. The chemical cost for the thickening aid was estimated as $1.38 per ton of dry solids. The other thickening aids investigated were ineffective.

b) An investigation of several filtration aids applied singly showed that the most effective in reducing specific resistance (24.8 x 10^7 sec ^{2}/g without filter aid) was cationic polymer Calgon M-510 (76% reduction at concentration of 1% by weight of solids). Other filtration aids were also effective in reducing specific resistance. To investigate the effect which might result by adding Calgon M-510 to a sludge which had been thickened using Calgon M-560, a series of filtration experiments in which the two polymers were added in equal amounts was conducted. A synergistic effect was found as the reduction in specific resistance by the combination was greater than the sum of the individual reductions. A reduction of 84% was observed at a total dosage of 0.5%.

c) Both Büchner test and leaf test data were used to estimate a vacuum filter loading rate of 10.45 lb/hr. ft^2 for dewatering a sludge thickened to 20% solids with 0.05% Calgon M-560 and further conditioned with 0.05% Calgon M-510. The filter cake solids content was estimated to be 49%. The cost of Calgon M-510 employed as a filtration aid amounted to $0.44 per ton of dry solids.

12. A possible system to collect, thicken, filter and dry the coagulated solids produced from the Republic Mine overflow was synthesized. In doing this, it was assumed that water quality requirements for reuse prohibited both the addition of coagulants within the reuse system and increasing the average percentage of process water recycled. It was estimated that this system would require an initial investment of $870,800. The average cost per ton of dry solids (including interest, depreciation, labor, thickening and filtration chemicals but excluding coagulation chemicals) was estimated as $37.65. It is questionable whether this system or any other system of comparable cost could be justified unless some economic value could be attached to the dried solids. In the absence of any economic value, the dry or semi-dry solids would probably have to be disposed of by burial. In this case drying would not be necessary and the solids processing cost would decrease to about $26.00 per ton exclusive of final disposal.

13. The synthesis of other alternative systems for dealing with the fine particulates contained in tailings basin overflows was hampered by:

    a) The lack of information on water quality requirements for reuse within ore concentrating processes.
    b) The lack of information which could be employed to predict any gradual build-up of important water quality parameters which may occur within a reuse system.

14. Settling column experiments in which the fine tailings particles contained in the Republic Mine basin overflow were diluted by factors of 5 and 10 with various natural waters showed that:

    a) An increase in temperature caused the effective settling velocities to increase. In certain instances, the concomitant decrease in viscosity accounted for less than half of the measured increase in settling velocity.
    b) Dilution generally tended to increase the effective settling velocities. As the dilution factor increased from 5 to 10, effective settling velocities increased by 0% to 50%.
    c) The effect of dilution was generally greater for natural waters higher in dissolved solids. The dilution effect was generally the greatest for a water containing a residual alum concentration. The relation between increase in effective settling velocity and diluent water dissolved solids content did not appear to be linear.
    d) The addition of 2 mg/l of alum generally increased the effective settling velocity by about 0 to 10%.

    Similar experiments conducted using Empire fine tailings particles showed less effect for changes in temperature, dilution and dissolved solids content of the diluent water. In fact, for Lake Superior water increasing the dilution from a factor of 5 to a factor of 10 caused solids removals to decrease by 2% to 10%. No consistent effect of diluent water dissolved solids content on effective settling velocity was observed. Likewise, no consistent effect of the addition of 2 ppm alum was observed.

15. One of the factors contributing to the difference in behavior between the Republic and Empire fine particles is believed to be the difference in particle size. Particle size distributions were determined for both the Republic and Empire fine tailings fractions using two methods; settling data and Stokes Law and direct measurement from photomicrographs. For the Republic tailings, the mean particle sizes (50% smaller by weight) determined by each method were nearly identical and equal to 1.3 microns. For the Empire particles, the optical method yielded a mean particle size

of 6.5 microns whereas the settling data yielded 8.0 microns. Generally, the optical method tended to estimate fewer numbers of very small and very large particles.

16. A field study was conducted to compare suspended solids removal predicted from column data with that actually occurring over 3,800 feet of the Republic Mine Tertiary Effluent Stream. The predicted removal was 20% whereas the actual removal was 24%.

SECTION II
RECOMMENDATIONS

1. Design of tailings storage and clarification systems should include a thorough hydrologic analysis of the proposed system. Particular attention should be given to the system water balance, storage capacity and clarification performance during the critical melt period.

2. More data should be gathered concerning the hydrology of other tailings impoundment systems so that expected quantities of water released during melt periods could be estimated with more confidence. Likewise, more data should be gathered concerning the hydraulic and sedimentation efficiencies of other tailings clarification basins.

3. More field data should be gathered concerning the transport of fine particles in natural water systems, particularly in thermally stratified impoundments.

4. A comprehensive investigation should be conducted to develop water quality criteria for reuse in ore concentrating processes. These criteria should be determined experimentally by pilot studies of various types of flotation and magnetic separation processes. It is generally recognized that the concentration of dissolved solids, particularly Ca^{++}, Mg^{++}, and other multi-valent cations may be significant.

5. An applicable methodology should be developed to predict any gradual build-up of important water quality parameters which may occur within a reuse system.

6. Potential uses of the dried solids resulting from the chemical coagulation operation at the Republic Mine should be investigated. Any economic value attached to this material would help to justify the possible solids handling system discussed in this report. In assessing the feasibility of any system for handling the coagulated solids, due consideration should be given to the results of studies similar to those recommended under Items 4 and 5 above. It may prove to be more economical to eliminate the overflow than to treat it and handle the resulting solids.

7. If a solids handling system similar to the one discussed in Section VI of this report is judged to be the optimal solution, a continuous flow pilot scale demonstration study should be conducted to develop design data for a full scale operation.

SECTION III
INTRODUCTION

Over the past 25 years, the iron mining industry has witnessed a trend toward the exploitation of low grade ore formations. Formerly, nearly all of the ore shipped consisted of richer, natural ores which required little or no beneficiation or concentration at the mine site. As the high grade ore deposits became scarcer, technological improvements in beneficiation processes made the exploitation of low grade ore deposits economically attractive. The term taconite, although descriptive of a hard, dense Minnesota rock with 25% to 35% iron content, has come to be frequently used to refer to low grade iron formations in general. In 1945, only 23% of the ore shipped resulted from beneficiation processes. However, in 1965, 77% of the 87 million tons of useable ore had been beneficiated. The Lake Superior district accounted for three-fourths of this production. It is anticipated that this trend will continue in the future.

In the concentrating process the low grade ore (25% to 35% Fe) is concentrated to 55% to 70% Fe. This results in a waste material containing 10% to 20% Fe. Thus, for every four tons of ore at 35% iron, 1.8 tons of concentrate at 65% iron would be produced. The 2.2 remaining tons at 10% iron would be discharged to the tailings deposit. The concentrated ore is pelletized in a heat hardening operation with the aid of a suitable binding material such as bentonite. The fact that this pelletized iron ore is an extremely desirable blast furnace feed is one of the factors responsible for the increased exploitation of low grade ore deposits.

A variety of unit operations such as gravity separation, screening, cycloning, magnetic separation and flotation are employed in the concentrating process. Nearly all of these operations are wet processes in which the ore materials are suspended in water. Naturally, considerable quantities of water are used in these processes. To produce one ton of concentrate from four tons of crude ore can require from 600 to 6,000 gallons of water depending on the process. Upon leaving the process, this water serves to transport the remaining waste materials to the tailings basins.

A system for handling tailings wastewater serves chiefly to separate the suspended load from the liquid so that the water can be either discharged to a natural watershed or reused directly. Since the tailings wastewater contains on the order of 70,000 to 500,000 ppm suspended solids (98% of which settles very rapidly), large volumes of solids are deposited. As pointed out above, the volume of this deposit is roughly equivalent to 50% to 75% of the volume of ore mined. In any tailings disposal system, therefore, a certain amount of land would be required for the permanent disposal of these solids. This has led to the development of

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huge tailings deltas in the proximity of concentrating plants. In some cases (the Reserve Mining Co. in Minnesota and some inactive Michigan copper mining operations), these deltas extend into Lake Superior. In beneficiation operations located further inland, these deposits may be better contained. Nevertheless, a substantial amount of supra-colloidal solids (500 ppm to 1,500 ppm) may remain in suspension and settle very slowly. Since most concentrating operations are located in areas of relatively low land values, the most economical solution has been to create large impoundments and to rely upon gravity sedimentation to accomplish separation of these solids.

One advantage of this impoundment system for secondary treatment is that the partially clarified wastewater may be reused in the plant operation. This reuse minimizes the quantity of pond effluent discharged into the natural water course and tends to make the plant water balance approximate a closed system. Naturally, this minimizes the cost of any treatment required for the pond effluent. However, in a particular situation, water reuse may be limited by the build-up of dissolved substances which may interfere with the concentrating process.

Some inherent problems associated with impoundment systems are caused by the areal extent of the ponds and their associated watersheds. Taken as a whole, these areas can be viewed as artificial watersheds which respond to meteorologic changes. The Lake Superior iron mining district receives a heavy snowfall; and, during the spring thaw, large quantities of water are released from the impoundments and their associated watersheds. Thus, even if water reuse is practiced, unless the impoundment has an adequate storage capacity, pond water may be discharged to the receiving stream at a high rate. Moreover, at high discharge rates accompanied by low water temperatures, the clarification capacity of the impoundment is impeded and the suspended solids content of the pond effluent tends to increase. Thus, in certain cases, further treatment may be required for the pond discharge.

The overall objective of this investigation was to advance and improve the applied technology related to the storage and disposal of wastewater resulting from the concentration of low grade iron ore, and, specifically:

A. To develop basin design and management criteria which recognize the influence of tailings settleability as well as the effects of meteorologic and hydrologic phenomena on the pond effluent quality.

B. To investigate alternative tertiary treatment methods for eliminating the fine particulate materials from the basin discharges with a view toward developing a tertiary treatment system for subsequent demonstration.

C. To investigate the physical and transport characteristics of the supra-colloidal particles contained in the tailings basin discharges with a view toward predicting the rate at which these particles might be removed from a natural water system.

These objectives were attained through a comprehensive study of the tailings impoundment systems associated with the iron ore concentrating plants associated with the Republic and Empire Mines, located in the Upper Peninsula of Michigan and operated by the Cleveland Cliffs Iron Company. In addition, complementary laboratory studies were carried out at the Sanitary Engineering Laboratory of Michigan Technological University. Subsequent sections of this report treat each aspect of this comprehensive study in detail.

At the outset of the study, it was also intended to study the impoundment system associated with the Humboldt Mine. However, midway through the study period, mining operations at the Humboldt site ceased and the concentrating plant was shut down for modifications. Therefore, the intended study of the Humboldt impoundment system was abandoned and is not included in this report.

SECTION IV
BASIN DESIGN AND MANAGEMENT CRITERIA

Hydrologic Aspect

As pointed out in the introduction, tailings basins and their surrounding areas can be viewed as vast artificial watersheds. An understanding of the hydrology of these watersheds is fundamental to the development of sound basin design and management criteria. To develop this understanding, water balances were formulated for the tailings basins associated with the Empire and Republic mines. In this way, the relative importance of precipitation (both rain and snow), snowmelt, evaporation, seepage and surface outflow could be judged.

Description of Study Areas

Climate and Meteorology

The two main study sites, the Republic Mine and the Empire Mine, are separated by approximately 15 miles, as shown in Figure 1. The two main river basins, the Michigamme near Republic and the Escanaba near Empire, drain into Lake Michigan. For this area, the mean monthly temperature ranges from 15°F in January to 66°F in July with an average annual temperature of about 41°F. The average date of the first fall temperature of 32° or colder is September 21 and the last day in the Spring is May 28. Average annual precipitation is about 31 inches with snowfall averaging about 110 inches. Snowfall during the 1970-71 study period totaled 135.5 inches with a maximum snow depth on the ground of 36 inches. Monthly evaporation potentials and a class A pan coefficient of 80% have been determined for this area by Wiitala *et al*. (1).

Two important factors affecting the magnitude and duration of spring runoff to be expected from tailings basins and their associated watersheds are the thaw duration and water equivalent of the snowpack at the beginning of thawing. A frequency analysis of the duration of the spring thaw (time elapsed between the onset of melting conditions and the disappearance of the snowpack) for this region based on 20 years of record (1951-1970) indicated a mean thaw duration of 22 days. The thaw duration was equal to or less than 39 days in 90% of the years and equal to or less than 12 days in 10% of the years. A similar analysis of the water equivalent of the snowpack at the onset of thawing conditions indicated a mean value of 8.5 inches. The snowpack water equivalent was equal to or greater than 6 inches for 90% of the years and equal to or greater than 11 inches for 10% of the years. Snowpack water equivalents did not appear to be correlated with thaw duration. The length of melt period and the water equivalent of the snow just prior to the melt for the 1970-71 study period were 15 days and 9.5 inches respectively. A water equivalent greater than this would be expected about once in three years. Likewise, a thaw duration shorter than this would be expected about once in 5 years.

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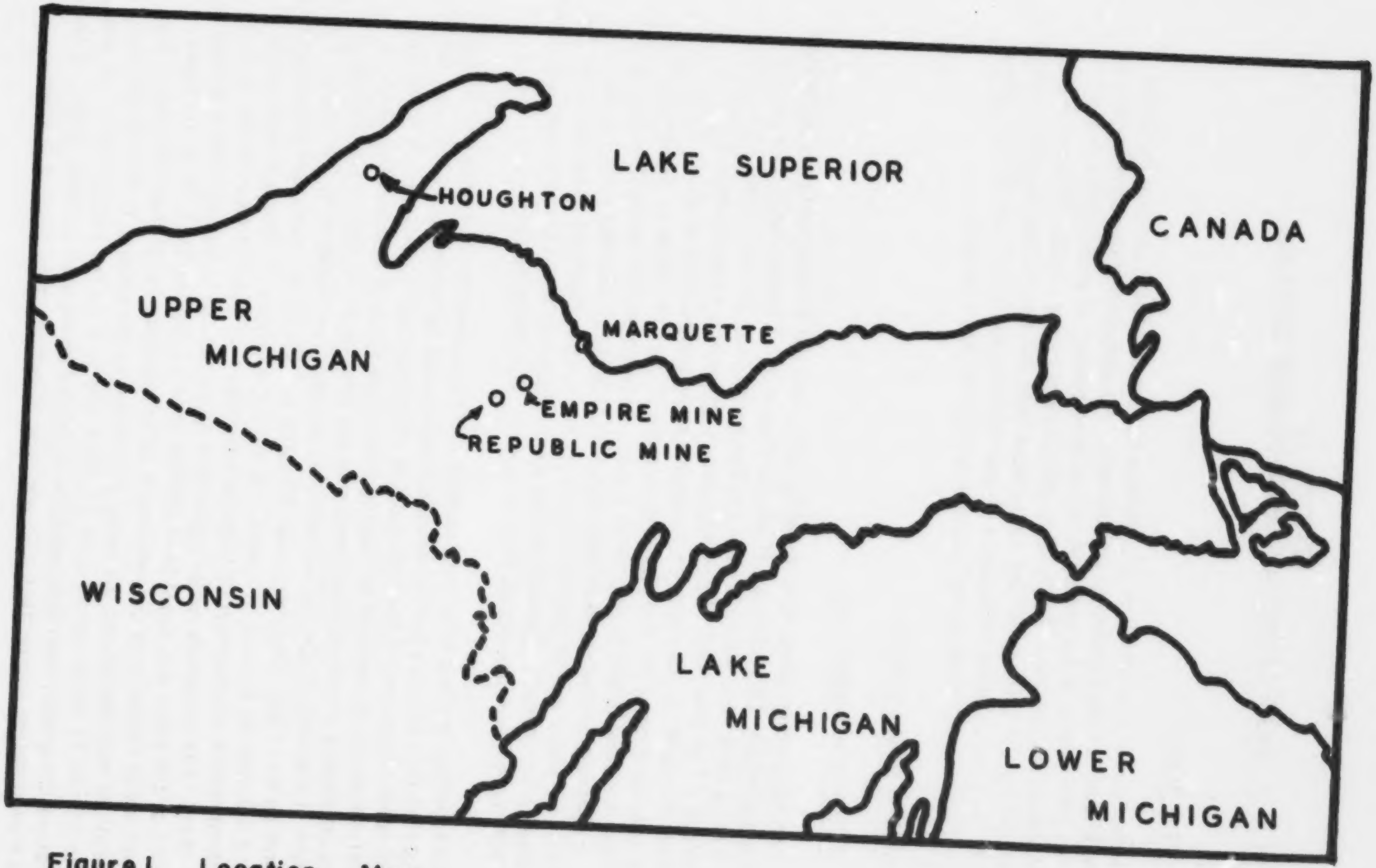
LAKE SUPERIOR
HOUGHTON
CANADA
UPPER
MICHIGAN
MARQUETTE
EMPIRE MINE
REPUBLIC MINE
WISCONSIN
LAKE
MICHIGAN
LOWER
MICHIGAN

Figure I. Location Map

Tailings Handling Operations at the Republic Mine

The Republic Mine is operated by the Cleveland Cliffs Iron Co (CCI) for CCI Jones & Laughlin Steel, Wheeling Steel, and International Harvester Companies and is located on the Marquette Iron Range approximately 30 miles southwest of Marquette, Michigan as shown in Figure 1. The Republic ore body consists of hematite (major economic material), magnetite, martite, quartz, jasper, iron silicates, and minor secondary carbonates. All of the constituents appear in the tailings deposits. The concentration plant processes roughly 20,700 long tons (2240 lb./long ton) per day of low grade hematite at 35.5% iron to produce roughly 9,700 long tons per day of concentrated ore at 65.5% iron. The remaining 11,000 long tons per day at roughly 10% total iron are discharged to the tailings basin. In the concentration process, the ore is first ground to a fine state (80% - 325 mesh). Argillaceous slime materials are then removed by wet cycloning. Subsequently, the concentrated ore is floated while the flotation underflows are discharged to the tailings stream. Thickening and vacuum filtration are employed to dewater the concentrated ore. This primary concentrate is then further concentrated in a regrind flotation process. The concentrate from the regrind operation is then dewatered and pelletized.

A simplified diagram of the concentration process and waste sources is shown in Figure 2. It can be seen from this figure that the waste arises from six principal sources. The total flow to the tailings basin amounts to roughly 59 cfs of which 58 cfs is water. Because of the supra-colloidal hematite particles in suspension, the tailings stream and ponds exhibit a red color similar to that of tomato juice.

The tailings slurry is conveyed in an open channel to the impoundment system. The Republic impoundment system covers an area of about 938 acres which is divided into 3 ponds and a return flow system. The average basin condition is about 30% water area and 70% tailings (land) area. The entire basin is enclosed by dikes and, in general, has an interior elevation higher than the surrounding land area. There is, therefore, no surrounding watershed. Figure 3 shows a plan view of the impoundment and clarification system. Pond 1 consists of a consolidated tailings deposit, and, including the Northeast Area, covers an area of 593 acres. The tailings stream spreads out and meanders over this deposit in sheet flows. During this process, the bulk of the solids settle out and become part of the consolidated deposit. The total annual tailings deposit, spread uniformly over the 593 acres of Pond 1, would cause an increase in the surface elevation of the deposit of approximately 3.5 to 4.0 feet. Spot checks of the tailings elevation at the beginning and end of the 378 day study period indicated that this was the case. Thus, it can be concluded that, over the study period, nearly all of the tailings deposit was stored in the Pond 1 area.

Pond 2 consists of an intermediate settling area of 78 acres. During the study period, the flow entered Pond 2 through a rock-fill dike. The water surface elevation is controlled by a weir structure ahead of the culverts leading to Pond 3.

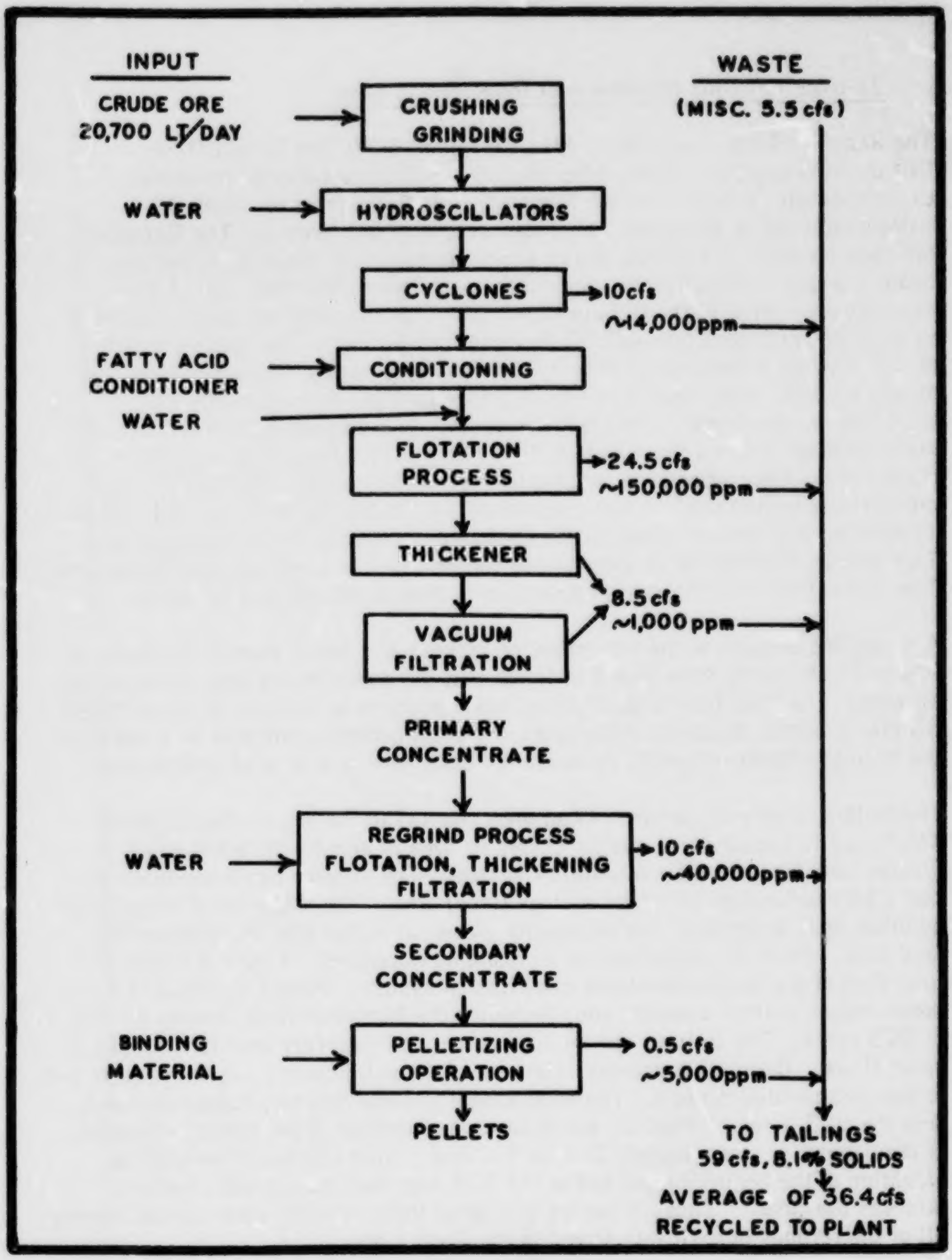
INPUT
WASTE
CRUDE ORE
20,700 LT/DAY
CRUSHING
GRINDING
(MISC. 5.5 cfs)
WATER
HYDROSCILLATORS
CYCLONES
10cfs
~14,000ppm
FATTY ACID
CONDITIONER
CONDITIONING
WATER
FLOTATION
PROCESS
24.5 cfs
~150,000 ppm
THICKENER
8.5 cfs
~1,000 ppm
VACUUM
FILTRATION
PRIMARY
CONCENTRATE
WATER
REGRIND PROCESS
FLOTATION, THICKENING
FILTRATION
10 cfs
~40,000ppm
SECONDARY
CONCENTRATE
BINDING
MATERIAL
PELLETIZING
OPERATION
0.5 cfs
~5,000ppm
PELLETS
TO TAILINGS
59 cfs, 8.1% SOLIDS
AVERAGE OF 36.4cfs
RECYCLED TO PLANT

Figure 2. Republic Mine, Simplified Concentration Process Diagram

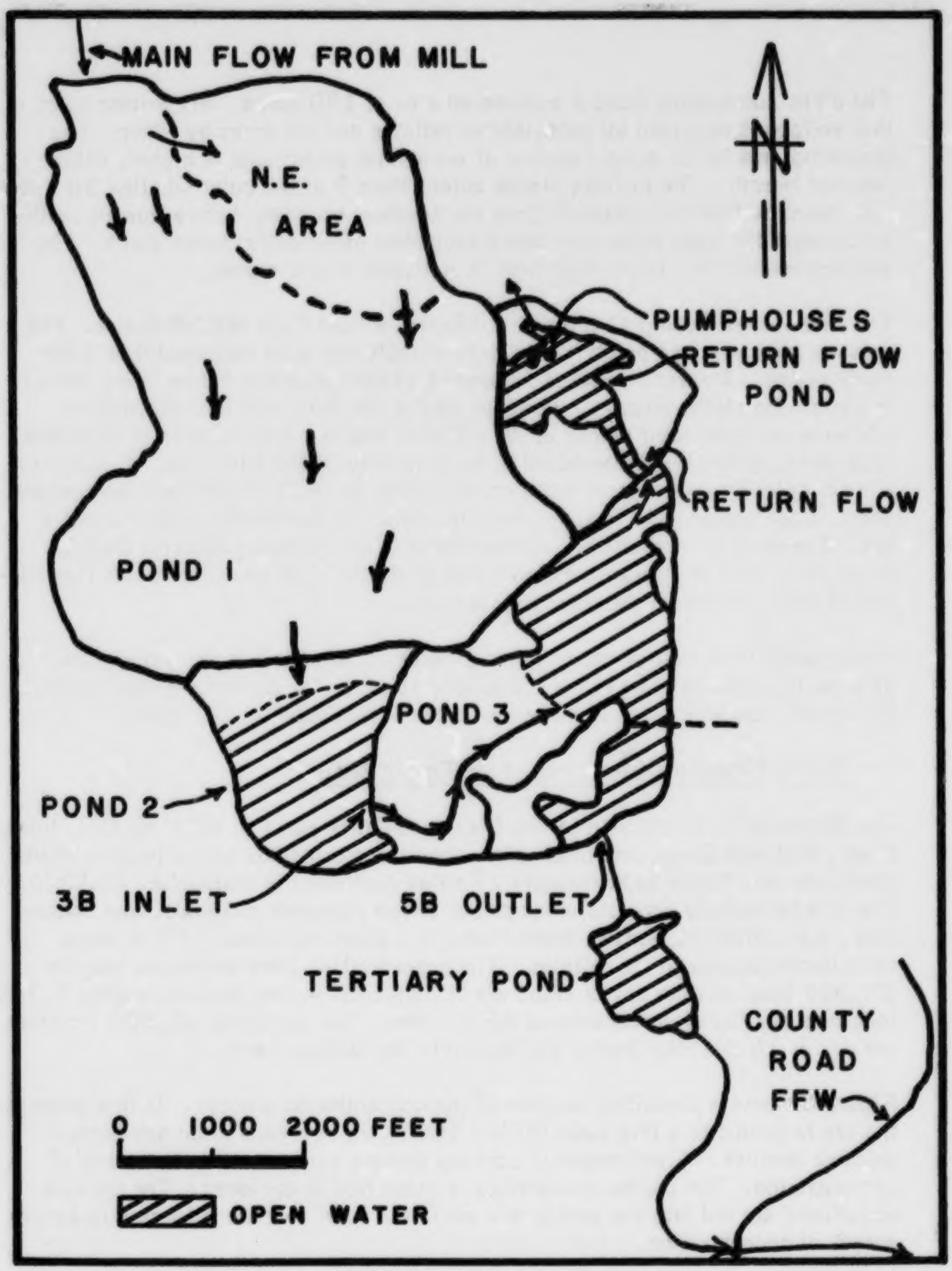
MAIN FLOW FROM MILL
N
N.E. AREA
PUMPHOUSES
RETURN FLOW POND
RETURN FLOW
POND 1
POND 3
POND 2
3B INLET
5B OUTLET
TERTIARY POND
COUNTY ROAD FFW
0 1000 2000 FEET
OPEN WATER

Figure 3. Republic Mine, Tailings Impoundment and Clarification System

The dikes surrounding Pond 3 enclose an area of 190 acres. Sixty-nine acres of this surface is occupied by consolidated tailings and 90 acres by water. The remaining area of 31 acres consists of peninsular projections of higher, natural forested terrain. The tailings stream enters Pond 3 at the point labelled 3B Inlet and tends to flow in a channel along the southern boundary before flowing northeast toward the open water area where secondary clarification takes place. The performance of this clarification area is analyzed in this report.

The flow leaves Pond 3 at two points labelled Return Flow and 5B Outlet. The flows at each of these points varied considerably and were measured during the study period. The return flow is pumped to a reuse reservoir before being reused in the concentrating process. The flow rate at the 5B outlet is controlled by adding or removing weir planks at the outlet structure. During times of high flow, alum and polyelectrolyte are added to the flow leaving the 5B outlet. Coagulation and flocculation takes place in the channel between the 5B outlet and the tertiary pond. Upon entering the tertiary pond, nearly all of the remaining solids settle out. The tertiary pond effluent is discharged to the receiving stream, Gambles Creek, which has an average upstream flow of 4 cfs. The coagulation and flocculation of this overflow is analyzed in this report.

A considerable amount of unconsolidated muck accumulated in the tertiary pond as a result of the settling of the coagulated solids. The thickening, dewatering and drying characteristics of these solids are investigated in this report.

Tailings Handling Operations at the Empire Mine

The Empire Mine is operated by the Cleveland Cliffs Iron Co. (CCI) for CCI, Inland Steel, McLouth Steel, and Internation Harvester Companies and is located on the Marquette Iron Range approximately 17 miles southwest of Marquette, Michigan. The Empire orebody consists of magnetite (major economic material), iron carbonates, iron silicates, martite, earthy hematite, chert and quartz. All of these constituents appear in the tailings. The concentration plant processes roughly 28,500 long tons per day of crude ore at 33% total iron to produce roughly 9,700 long tons per day of concentrate at 66.5% iron. The remaining 18,800 long tons per day at 16.5% total iron is discharged to the tailings basin.

Figure 4 shows a simplified diagram of the concentration process. In this process, the ore is ground to a fine state (90% - 500 mesh) by means of an autogenous grinding process. A wet magnetic cobbing process serves as the first stage of concentration. The cobber concentrate is classified in cyclones. The cyclone underflows are fed into the pebble mill while the overflows are subjected to further stages of concentration.

The siphonizer serves to dewater and deslime the cyclone overflows thereby upgrading the iron content. The final concentration steps consist of further magnetic concentration and an amine flotation. The process stream is then

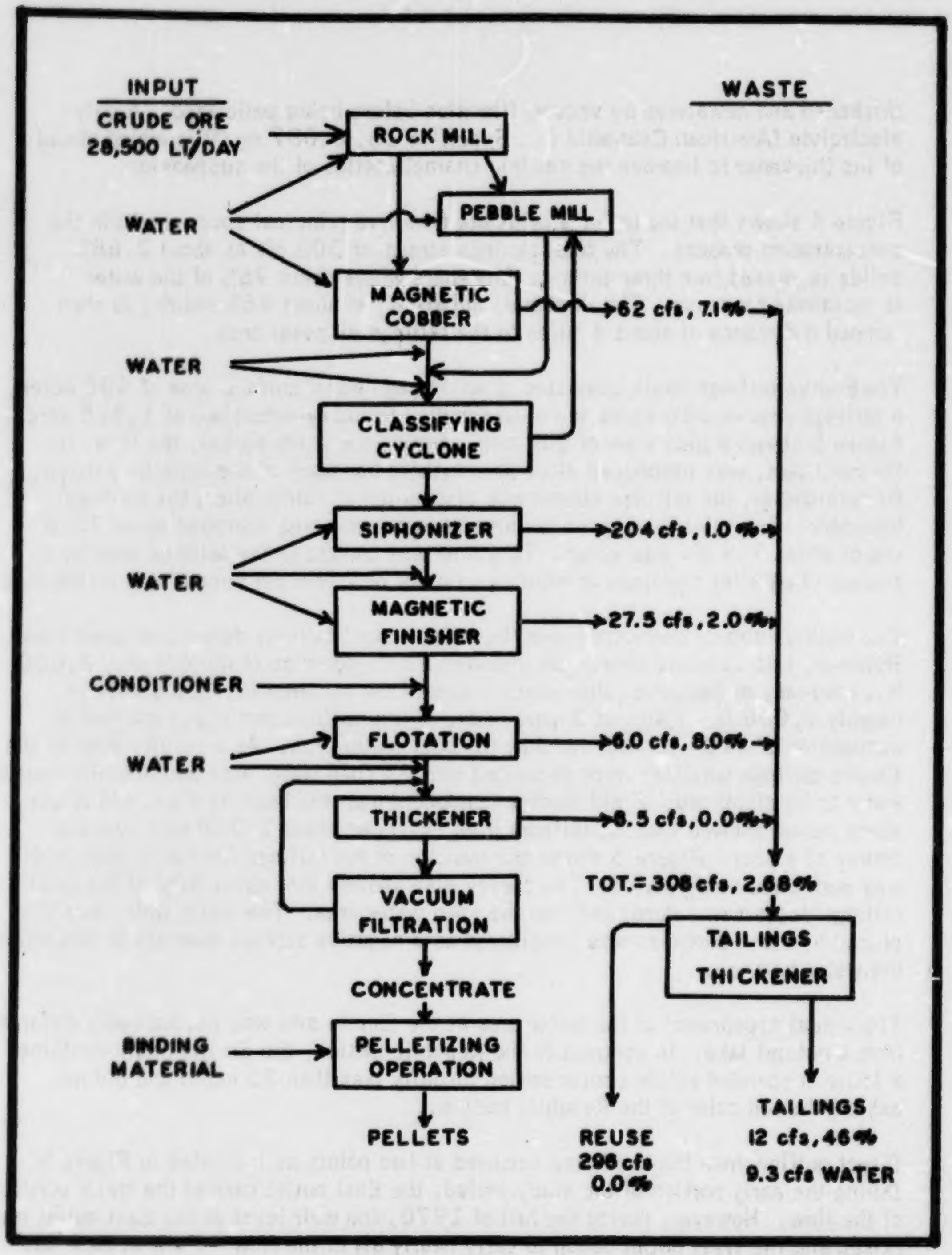
INPUT
CRUDE ORE
28,500 LT/DAY
ROCK MILL
PEBBLE MILL
WATER
MAGNETIC COBBER
62 cfs, 7.1%
WATER
CLASSIFYING CYCLONE
SIPHONIZER
204 cfs, 1.0%
WATER
MAGNETIC FINISHER
27.5 cfs, 2.0%
CONDITIONER
FLOTATION
6.0 cfs, 8.0%
WATER
THICKENER
8.5 cfs, 0.0%
VACUUM FILTRATION
CONCENTRATE
BINDING MATERIAL
PELLETIZING OPERATION
PELLETS
WASTE
TOT.= 308 cfs, 2.68%
TAILINGS THICKENER
REUSE
296 cfs
0.0%
TAILINGS
12 cfs, 46%
9.3 cfs WATER

Figure 4. Empire Mine, Simplified Concentration Process Diagram

thickened and dewatered by vacuum filtration before being pelletized. A polyelectrolyte (American Cyanamid Co. Superfloc 16, 0.087 mg/l) is added ahead of the thickener to improve the settling characteristics of the suspension.

Figure 4 shows that the tailings originate from five principal sources within the concentration process. The total tailings stream of 308 cfs at about 2.68% solids is passed into three tailings thickeners where about 96% of the water is reclaimed for reuse. The thickener underflow, at about 46% solids, is then pumped a distance of about 5 miles to the tailings disposal area.

The Empire tailings basin consisted of an average water surface area of 406 acres, a tailings area of 326 acres and a surrounding tributary watershed of 1,315 acres. Figure 5 shows a plan view of this area. During the study period, the flow, for the most part, was introduced along the northern boundary of the tailings deposit. Occassionally, the tailings stream was discharged at points along the southern boundary. The total flow to the tailings impoundment area averaged about 12.8 cfs of which 9.9 cfs was water. This flow was carried to the tailings area by a system of parallel pipelines in which a velocity of 7 feet per second was maintained.

The tailings stream meandered over the consolidated tailings deposit in sheet flows. However, this overland sheet flow occurred over a distance of roughly only 2,000 ft.; whereas, at Republic, the overland sheet flow occurred over a distance of roughly 6,000 ft. Thus, at Empire, more of the tailings particles remained in suspension when the tailings reached the open water area. As a result, most of the Empire tailings particles were deposited into the open water area and actually caused water to be displaced. Field studies conducted near the beginning and end of the study period showed that the tailings front advanced about 2,000 feet over the course of a year. Figure 5 shows the position of the tailings front at a point midway during the study period. The survey also showed that about 80% of the total tailings deposit was deposited into the open water area. The water which was displaced by these particles was considered as a negative storage quantity in computing the water balance.

The visual appearance of the water area at the Empire site was not markedly different from a natural lake. In contrast to the Republic basins, the Empire basin contained a lower suspended solids concentration (usually less than 25 mg/l) and did not exhibit the red color of the Republic basins.

Direct outflow from the pond area occurred at two points as indicated in Figure 5. During the early portion of the study period, the East outlet carried the major portion of the flow. However, during the fall of 1970, the weir level at the East outlet was raised and the West outlet began to carry nearly all of the flow. Flow at each outlet was measured during the course of the study.

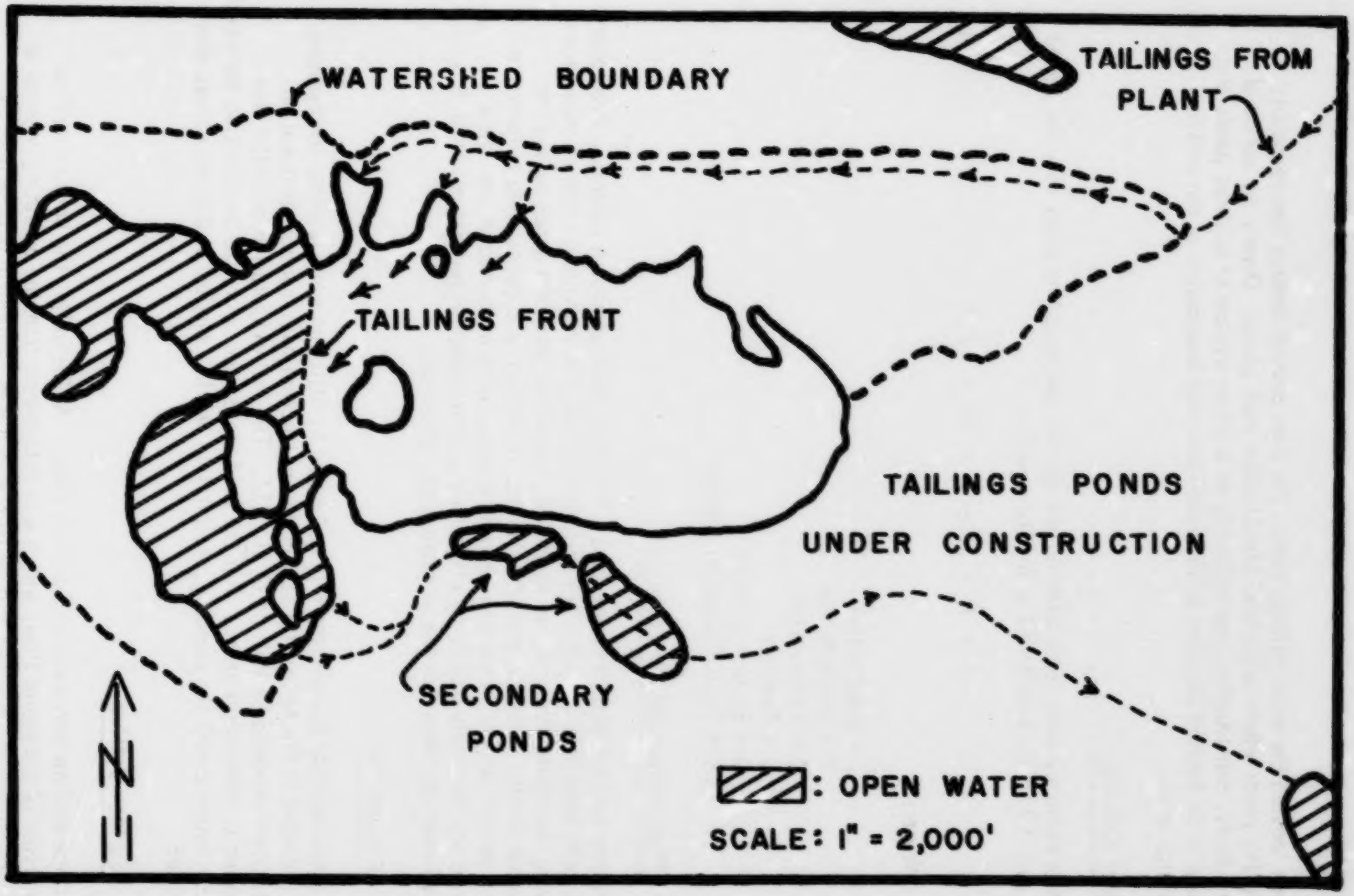
WATERSHED BOUNDARY
TAILINGS FROM PLANT
TAILINGS FRONT
TAILINGS PONDS UNDER CONSTRUCTION
SECONDARY PONDS
▨ : OPEN WATER
SCALE: 1" = 2,000'
N

Figure 5. Empire Mine, Tailings Impoundment and Clarification System

Upon leaving the main tailings basin, the flow passed through two secondary settling ponds where additional clarification took place. During the period of this study, construction was underway on a larger system of tailings ponds and dikes to be located adjacent to the south and east boundaries of the existing tailings area.

Water Balances

Water balances were formulated for the Empire and Republic sites over the period of July 1970 to June 1971 using the equation

$$I + P - E - L = Q + \Delta S \qquad 1.$$

Where:

I = Plant Inflow
P = Precipitation
E = Evaporation
L = Seepage
Q = Surface Outflow
ΔS = Total Change in Storage

Plant Inflow and Precipitation

Monthly average plant inflows were taken from plant operation records. Precipitation was measured at both sites utilizing standard 8 inch non-recording rain gages during the entire study period as well as recording type rain gages during the summer months. During periods when the recording type gages were inoperative, long period catches in the non-recording gages were broken down into their corresponding daily values by utilizing the climatological records of the U.S. Environmental Data Service at Marquette, Michigan.

Evaporation

Average monthly evaporation was estimated by utilizing mean monthly temperature as recorded at the Marquette station in conjunction with monthly evaporation potentials reported by Wiitala et al. (1). The full evaporation potential was applied to open water areas. For the land areas, however, the evaporation potential was compared with the monthly precipitation and the smaller of the two values was applied.

Surface Outflow

Surface outflow was measured at both sites. At Republic, the surface outflow consisted of the return flow, which was determined from the operating records of calibrated pumps, and the 5-B outflow which was continuously measured by recording the water elevation above a weir. At Empire the surface outflow was continuously measured by recording the water elevation ahead of hydraulic control sections.

Storage

The total change in storage can be expressed as

$$\Delta S = \Delta S_p + \Delta S_s + \Delta S_i + \Delta S_g + \Delta S_t \qquad 2.$$

Where:

ΔS = Total Change in Storage
ΔS_p = Change in Pond Water Storage
ΔS_s = Change in Snowpack Storage
ΔS_i = Change in Tailings Ice Storage
ΔS_g = Change in Groundwater Storage
ΔS_t = Change in Tailings Storage

The change in pond water storage, ΔS_p, was determined by multiplying the recorded changes in pond water surface elevations by the pond areas. This quantity also reflected the volume of ice and snow floating on the pond surface. In cases where solid tailings were deposited into open water a negative storage correction was applied to account for the pond water displaced by the tailings.

Tailings storage, S_t, refers to the interstitial water stored within a saturated tailings deposit as it builds up. This was determined by multiplying the change in volume of saturated tailings by the average porosity.

Snowpack storage, S_s, was determined from periodic snow surveys conducted at the sites during the winter months. Snow depths and water equivalents were measured on both the tailings deposits and surrounding watersheds. The samples taken on the tailings deposits did not include any tailings ice which existed under the snow. Tailings ice refers to ice formed on the tailings deposit as a result of the meandering overland sheet flows. Winter accumulations of tailings ice storage, S_i, were estimated by closing the water balance.

Groundwater storage refers to subsurface water stored in the natural watershed tributary to the tailings system. No field measurements of changes in ground water storage were made. Since the study commenced during the summer and lasted 1 year it was assumed that the ground water storage at the beginning of the study period was equal to that at the end of the study period. Likewise, no snow or ice were present at the beginning and ending dates. Thus over the entire study period:

$$\Delta S_s = 0 \qquad 3.$$

$$\Delta S_i = 0 \qquad 4.$$

$$\Delta S_g = 0 \qquad 5.$$

and $$\Delta S = \Delta S_p + \Delta S_t \qquad 6.$$

Seepage

Since there was no practical way to directly measure seepage, total seepage during the study period was estimated by closing the water balance. Combining Equations 1 and 6 gives:

$$L = (I+P-E) - (\Delta S_p + \Delta S_t + Q) \qquad 7.$$

which was used to estimate total seepage during the study period.

At Empire, no drastic changes in basin configuration took place during the course of the study. Thus, it was assumed that seepage occurred at a constant rate throughout the year. At Republic, however, flow was diverted into the northeast portion of Pond 1 at the beginning of April 1971. This visibly increased the opportunity for seepage along the northeast boundary. Therefore, it was assumed that seepage occurred at a constant rate over the period of June 15, 1970 to March 31, 1971 and at another, constant, but greater rate, over the period of April 1, 1971 to June 28, 1971. The first rate was estimated by applying Equation 7 over the period from June 15 to October 31, 1970. In developing Equation 7, it was assumed that $\Delta S_g = 0$. Fluctuations of ground water stage in this area indicate that, over this period, ΔS_g would be negative causing this seepage rate estimate to be somewhat low (2). The second rate was estimated by applying Equation 7 on the period of April 1 to June 28, 1971. In this case, the assumption that $\Delta S_g = 0$ would cause the seepage rate estimated to be somewhat high.

Water Balance Results

Tables A1 and A2 (Appendix A) show the water balance data for Empire and Republic respectively. The basic volume measure employed here is the second-foot-day (sfd) which is equal to the volume accumulated if 1 cfs were to flow for 1 day, or 8.64×10^4 cubic feet. These data are summarized by the hydrographs and mass curves shown in Figure A.1 to A.3 (Appendix A). Combining Equations 1 and 2 gives:

$$\Delta S_s + \Delta S_i + \Delta S_g = (I+P-E-L) - (Q + \Delta S_p + \Delta S_t) \qquad 8.$$

Thus the quantity of water stored as snow, tailings ice, and groundwater is shown by the difference in ordinates between the Inflow + Precipitation - Evaporation - Seepage line and the Outflow + Storage line on the mass curve. This quantity is important because it represents an additional volume of water released during the spring thaw.

Table 1. Comparison of Annual Water Budgets for Republic and Empire Tailings Systems Based on Unit Areas of the Tailings System

	Republic	Empire
Area of Tailings System sq. mi.	1.46	1.14
Area of Surrounding Watershed sq. mi.	0.0	2.06
Plant Inflow, I, inches	542	119
Precipitation, P, inches	37	107
Evaporation, E, inches	-27	-67
Net Input, I+P-E, inches	552	159
Direct Surface Outflow, Q, inches	474	73.4
Seepage, L, inches	60	93
Change in Storage, $\Delta S_p + \Delta S_t$, inches	+18	-7.4
Net Outflow + Change in Storage, $Q+L+\Delta S_p+\Delta S_t$, inches	552	159

Table 1 shows a comparison of the Republic and Empire water budgets based on unit areas of the impoundment system. The precipitation and evaporation figures include the volumes contributed by the tributary watershed. However, all the figures are based on unit areas of the impoundment system. The quantities, given in inches, represent the height of a volume of water spread over the area of the tailings basin. Thus, the annual plant inflow to the Republic basin amounted to a volume of water 542 inches high spread over 1.46 sq. miles; whereas the annual Empire plant inflow amounted to a column of water only 119 inches high spread over 1.14 sq. miles. These large differences in annual loading between the two systems are partially a result of the fact that water is reused from the Republic basins.

Because the direct hydraulic loading of the Empire basin is so much less, precipitation, evaporation and seepage play a much larger role in its overall water balance. Moreover, the importance of precipitation is accentuated by the 2.06 sq. mi. of tributary watershed at Empire. Table 2 shows the relative

Table 2. Relative Importance of Precipitation, Evaporation Storage and Seepage

		Republic	Empire
Percent of Annual Inflow	Plant Inflow	98.1%	75%
	Precip. less Evap.	1.9%	25%
Annual Change in Storage as Percent of Inflow		3.3%	-4.7%
Percent of Annual Outflow	Direct Surface Outflow	87.7%	44.4%
	Seepage	11.3%	55.6%

Table 3. Comparison of Maximal Tailings Ice and Snow Storage Based on Unit Areas of the Tailings Impoundments

	Republic	Empire
Unaccounted Storage, $\Delta S_i + \Delta S_s + \Delta S_g$ inches, water equiv.	15.5	25.8
Snow Accumulation, ΔS_s inches, water equiv.	1.8	15.7
Accumulation of Tailings Ice and Ground Water Storage, $\Delta S_i + \Delta S_g$ inches, water equiv.	13.7	10.1

importance of precipitation and seepage at each site. Precipitation less evaporation accounted for only 1.9% of the total input to the Republic basin, but amounted to 25% of the total input to the Empire basin. Likewise, seepage accounted for only 11.3% of the outflow at Republic, but made up 55.6% of the outflow at Empire. On a unit area basis, the seepage rates compared much better as the 93 in./yr. of seepage at Empire was only about 1.5 times the 60 in./yr. at Republic. It would be expected that the seepage be greater at Empire since the natural terrain underlying the tailings deposit dips to the southeast at roughly 20 to 30 feet per 1,000 feet. Thus, the dike along the southeast boundary of the impoundment reaches a height of about 90 feet. The natural terrain underlying the Republic basin is generally flatter, and except for one small section along the west boundary, the dikes are generally less than 30 feet in height.

Table 3 shows the maximal accumulations of tailings ice and snow. These quantities are given in inches and are based on the areas of the tailings impoundments (938 acres at Republic and 726 acres at Empire) exclusive of any tributary natural watershed. The unaccounted storage represents the additional amount of water (above and beyond the average flow) which was released during the spring melt. The snow accumulation represents the portion which existed as snow on the impoundments and/or on the surrounding natural watershed. Subtracting the snow accumulation from the total unaccounted storage gives the accumulation of tailings ice and ground water storage.

At Empire, the 25.8 inches of unaccounted storage, or additional water released during the spring melt, was significanly greater than the 15.5 inches at Republic. This was caused by the heavy snow accumulations on the 1,315 acres of surrounding watershed. Republic had no tributary watershed surrounding the diked area and accumulated only 1.8 inches of water equivalent as snow on the impoundment. In the absence of a surrounding natural watershed, ground water storage may be neglected. Thus, the remaining 13.7 inches represents the accumulation of tailings ice.

A study of Tables A.1 and A.2 and the hydrographs, mass curves and temperature data given in Figures A.1 and A.4 (Appendix A) shows that, at Republic, the spring melt begain in early March and was essentially completed by late April, only a few days after the snow was gone. This is what would be expected in the absence of ground water movement into the impoundment system.

During March, the average daily air temperatures seldom exceeded 32°F and the 10 day mean temperatures were consistently below 30°F. Therefore, it appears that the melting conditions may have been caused by a combination of condensation and the adsorption of solar radiation by the dark colored deposits of tailings.

A study of the same figures reveals that the Empire site behaved quite differently during the melt period. The surface outflow rate increased markedly at the beginning of April when the 10 day mean air temperature exceeded 32°F. Moreover, the high outflow rate continued through June even though the snow was essentially gone by April 20. This is indicative of a substantial amount of subsurface water movement into the impoundment system.

The interpretation of the hydrographs shown in Figures A.1 and A.2 (Appendix A) is complicated by the fact that changes in storage were also occurring. It is estimated that, in the absence of storage, the 10 day mean surface outflow at Republic would have reached 84 cfs during early April 1971. However, a considerable amount of water (252 sfd) was stored at Republic between April 4 and April 10 as a result of the diversion of the tailings stream into the Northeast Pond 1 area on April 4. This is a good example of how prudent management can mitigate the effect of peak flows. Likewise, at Empire, it is estimated that in the absence of storage, the 10 day mean flow would have reached 50 cfs.

Table 4 summarizes the peak 10 day mean outflows at the two sites. A comparison of the ratios of peak 10 day mean flows to average flows shows that the effect of the high spring flows is relatively much greater at Empire than at Republic. The incremental flow rates (computed peak 10 day mean flow in the absence of storage minus the average annual flow) were more comparable. Expressed on a unit area of impoundment basis, the incremental flow rates amounted to 22.6 and 38.5 cfs/mi^2 for Republic and Empire respectively. The higher value for Empire reflects the contribution of the tributary watershed.

Performance of Tailings Clarification Basins

The development of sound basin design and management criteria requires an understanding of both basin hydrology and clarification performance. It is essential to estimate both quantities of water to be handled and effluent quality attained by these basins. Information presented in the preceeding paragraphs can be used with engineering judgement as a guide to estimate water quantities.

In general, the quality of a water is determined by temperature and by the concentration of various dissolved and suspended materials present. The major parameter affecting the quality of tailings basin effluents is suspended solids concentration. In certain cases, temperature and the concentrations of dissolved organic and inorganic substances may appreciably affect water quality. For the purposes of this report, tailings pond effluent quality is judged by suspended solids concentration. Thus, the objective of this research aspect was to develop a method, based on laboratory settling data, by which the suspended solids concentration of a tailings basin effluent could be predicted and to evaluate this method by comparing predicted and actual concentrations obtained from operating tailings clarification basins.

A brief description of the research plan employed to achieve this objective is as follows:

Republic Mine

The Republic tailings suspension settled as a classical flocculent suspension. The settling characteristics of the waste were quantitatively described through the use of quiescent settling column tests (3). From the data developed in these tests, it was possible to predict the effluent concentration for an ideal plug flow basin.

To better predict the performance of the actual basins, residence time distributions were determined by using dye tracer techniques. By combining the residence time distribution data with the settling column data, it was possible to predict effluent concentrations for the actual basins. In order to compare predicted and actual concentrations, suspended solids concentrations were routinely measured at various points within the Republic tailings system during the course of the study. Periods of approximately steady state operation were selected from these field data. Knowing the flow rates, temperature, and mean residence times during these periods of pseudosteady-state operation, it was possible to predict the basin effluent suspended solids concentration from quiescent settling column data gathered under similar conditions.

Empire Mine

Unlike the Republic tailings, the Empire tailings with an initial solids content of 46% did not exhibit flocculent settling. Almost immediately after the start of each column test, an interface was formed and hindered settling took place. Because flocculent settling did not occur, the settling curve analysis employed for the Republic tailings could not be applied here. The supernate above the interface contained less than 5 mg/l of suspended solids whereas the effluent from the actual pond operating at an overflow rate of approximately 1% of the interface subsidence rate ranged from 10 to 20 mg/l. In this case, the suspended solids concentration remaining within the supernate could be used to formulate a rough estimate of that in the pond effluent. It is likely that the concentration increment in excess of that obtained in the column test was caused by wind action and scour at high flows.

Quiescent Settling Tests: Performance of an Ideal Plug Flow Basin

Methodology and experimental conditions employed in performing the quiescent settling tests are described in Appendix B. A total of 17 individual column tests employing the Republic tailings were conducted at various temperatures (40°F to 75°F), initial suspended solids concentrations (265 mg/l to 12,600 mg/l) dissolved solids content (114 mg/l to 150 mg/l) and coagulant dosages. The dissolved solids content of the tailings stream varied from day to day and was carefully noted since settleability had been observed to be a function of dissolved solids (4). A wide range of conditions was investigated so that comparisons could be made between the column data and the field data.

The results of the settling tests can be conveniently represented by noting the measured concentrations on a plot depth versus time. Figure 6 shows the results of a typical settling test. The curves represent iso-concentration lines located by interpolation between the measured concentrations. For discrete settling, the iso-concentration lines would be linear. Curvilinear iso-concentration lines indicate the flocculent nature of a suspension (3).

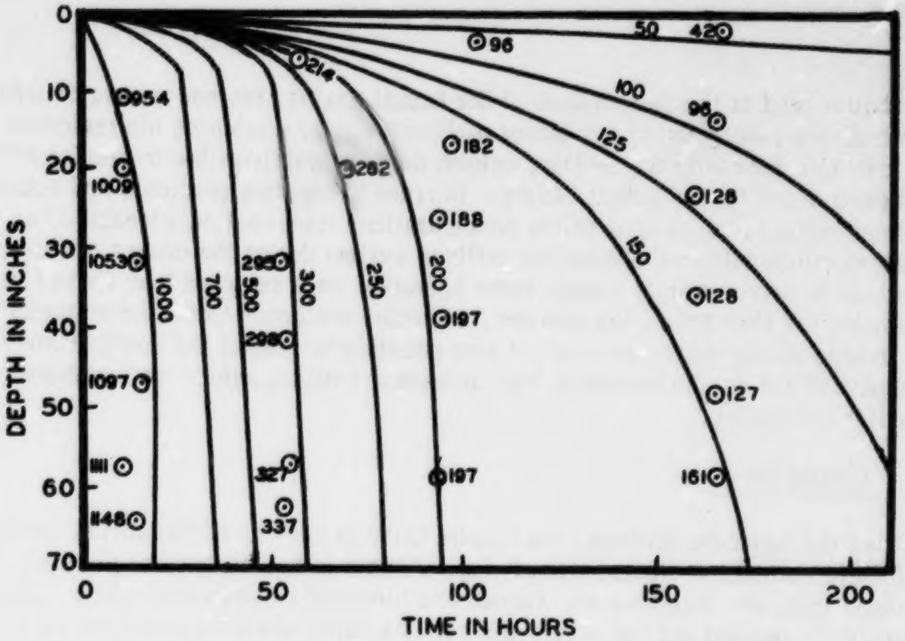

Figure 6. Results of Typical Quiescent Settling Test, Republic Tailings.

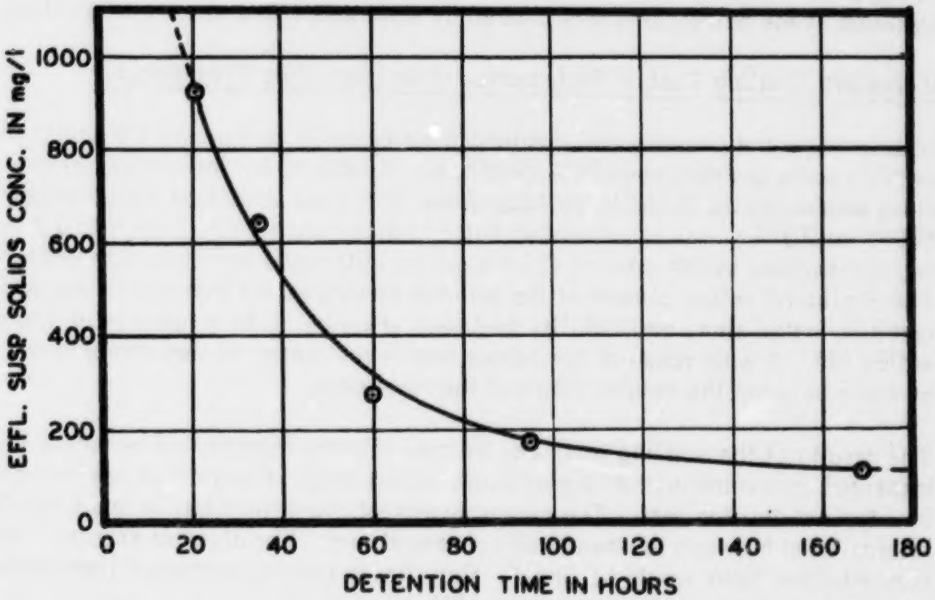

Figure 7. Typical Performance Curve for an Ideal, Plug Flow Basin

Table 4. Summary of Peak, 10 Day Mean Direct Surface Outflows

	Republic	Empire
Average Annual Flow, cfs	51	6.15
Actual Peak - 10 Day Mean	60	20.2
Computed Peak - 10 Day Mean Assuming No Storage, cfs	84	50
Ratio of $\frac{\text{Actual Peak - 10 D.M.}}{\text{Average Annual}}$	1.18	3.28
Ratio of $\frac{\text{Computed Peak - 10 D.M.}}{\text{Average Annual}}$	1.65	8.15
Incremental Flow, cfs (Computed Peak 10 D.M. - Ave.)	33	43.9
$\frac{\text{Incremental Flow Rate}}{\text{Unit Area of Impoundment System}}$ $\frac{\text{cfs}}{\text{sq.mi.}}$	22.6	38.5

The iso-concentration lines constructed from a quiescent settling column test can be used to predict the overall removal to be expected in a continuous flow basin under ideal plug flow conditions. Ideal plug flow conditions exist when every part of the fluid entering the basin is evenly distributed over the entire vertical cross section of the entrance zone, and the flow advances at a uniform and constant horizontal velocity to the outlet zone. Such an hypothetical continuous flow settling basin will be termed an "ideal basin". The procedure employed in predicting effluent concentrations as a function of detention time for an ideal basin of a particular depth is developed and explained in standard reference works (3). Figure 7 was developed from the data given in Figure 6 and shows the effluent concentration for an ideal basin 57 inches deep (average depth of Republic Pond 3) as a function of detention time.

Residence Time Distributions: Performance of a Non-Ideal Basin

Application of the foregoing analysis of an ideal basin to an actual situation requires consideration of the factors which affect settling in an actual basin. The efficiency of the sedimentation operation is largely a function of the hydraulic characteristics of the basin in which the operation takes place. The term hydraulic characteristics as used in this report refers to the flow pattern produced in the basin. This flow pattern or regime is related to the physical features of the basin such as length, width, depth, inlet and outlet conditions. The prevailing flow pattern in a basin determines the hydraulic efficiency of that particular basin. The term hydraulic efficiency as used here refers to the residence time distribution of the fluid in the basin. A basin's flow regime seldom consists of only one type of flow, but is usually a combination of several types ranging from idealized plug flow at one extreme to a completely mixed pattern on the other depending on the hydraulic characteristics of the basin.

The flow regime and residence time distribution of an actual basin can be determined both qualitatively and quantitatively by use of tracing methods. A tracer is introduced into the influent to the basin and its concentration at the outlet is determined as a function of time. From the results of a tracer test employing a one shot injection of tracer, a curve of effluent tracer concentration versus time may be plotted. The shape of the dispersion curve is a qualitative measure of the flow pattern through the basin. The curve itself can be interpreted as a residence time distribution of the fluid in the basin. If the dispersion curves are plotted in dimensionless terms, they can be used to compare the hydraulic characteristics of basins with different shapes or of the same basin with different flow rates.

In order to determine the residence time distributions, tracer studies were performed on both Pond 3 and the Tertiary Pond at Republic and on the main tailings pond at Empire. In each case, a slug of fluorescent tracer was injected into the influent and the concentration in the effluent stream was measured as a function of time. The methodology employed in these studies is described in Appendix C.

The dispersion curve for the northern portion of Republic Pond 3 is shown in Figure 8. During the time of the dispersion test, the average return flow rate was determined to be 36 cfs and the flow rate at 5B to be 2.2 cfs. In view of the pond configuration shown in Figure 3, it is reasonable to consider the northern 75 percent of the pond area (above the dashed line in Figure 3) as carrying the return flow, and the southern portion as carrying the 5B overflow. With this assumption the theoretical detention times for the northern portion and the southern portion were determined to be 117 and 545 hours, respectively. Since the majority of the flow was in the direction of the return flow outlet the dispersion curve for the northern portion, measured at the return flow outlet was of primary interest. The irregular shape of the declining portion of the curve shown in Figure 8 was very likely caused by a portion of the dye being trapped in a small pond at the tailings front and later being released.

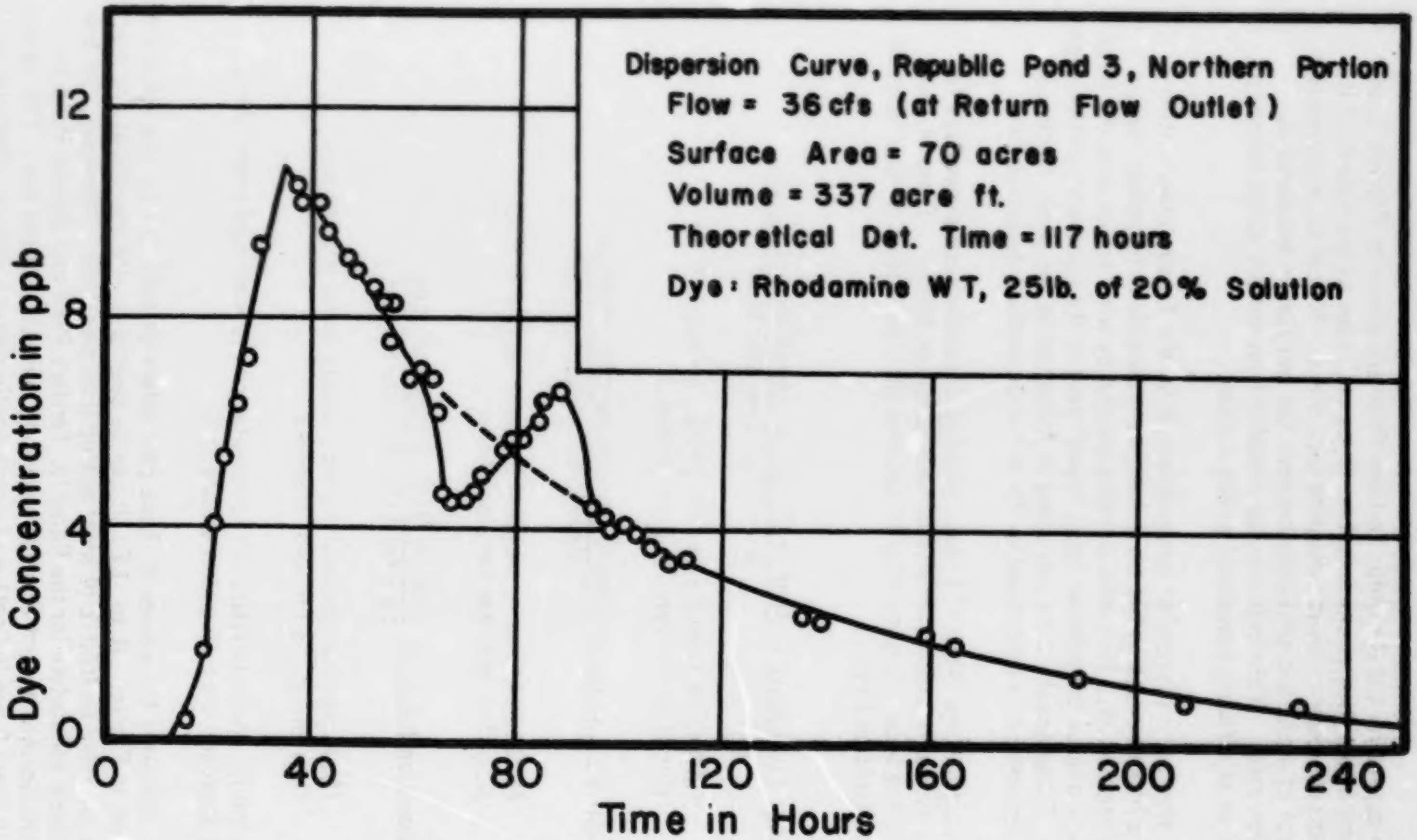

Figure 8. Dye Dispersion Curve for Republic Pond 3, Northern Portion.

Dispersion curves for the Republic Tertiary Pond are shown in Figures 9 and 10. In view of the theoretical detention times indicated on the figures it is evident that considerable short circuiting took place. Moreover, a comparison shows a lack of geometric similarity between the two curves measured at different flow rates. This indicates an unstable flow pattern which hampered the prediction of effluent suspended solids content.

Figure 11 shows the dispersion curve observed for the Empire clarification basin. The irregularities in the curve here were considerably greater than in the preceeding curves. This was probably caused by wind action over the large water surface and by the relative large "dead" area in the northwest portion (see Figure 5). Considerable short circuiting is indicated as the peak, tracer concentration occurred at a time equal to 7% of the theoretical detention time.

Various investigations (5)(6)(7) have proposed dimensionless parameters for judging the hydraulic characteristics of sedimentation basins. Three of these parameters have been employed here to compare the hydraulic characteristics of tailings clarification basins.

$$\text{Initial Time Ratio} = \frac{\text{time to the initial appearance of tracer}}{\text{theoretical detention time}}$$

(Measures severe short circuiting, equals 1.0 for plug flow and zero for ideal mixing.)

$$\text{Modal Time Ratio} = \frac{\text{time to peak tracer concentration}}{\text{theoretical detention time}}$$

(Measures dead or stagnant regions, equals 1.0 for plug flow and zero for ideal mixing.)

$$\text{Dispersion Index} = \frac{\text{time for 90\% of tracer to pass}}{\text{time for 10\% of tracer to pass}}$$

(Measures longitudinal mixing, equals about one for plug flow and 23.6 for ideal mixing.)

Optimal hydraulic characteristics for sedimentation are indicated when these parameters approach their plug flow values.

Table 5 compares the values of these parameters calculated from the dispersion curves shown in Figures 8 to 11. It can be seen that both the initial and modal time ratios are more indicative of ideal mixing than plug flow. Moreover, the low values of these parameters for the Republic Tertiary Pond and Empire Pond indicate a considerable amount of short circuiting and stagnant regions. The values for Republic Pond 3 are indicative of significantly better hydraulic characteristics than the others. In view of the plan configuration shown in Figures 3 and 5,

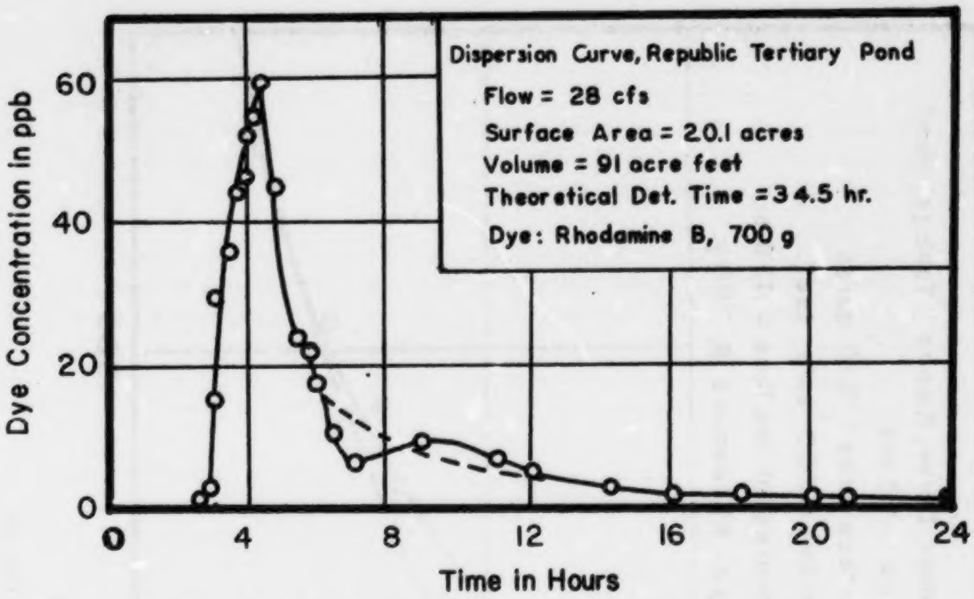

Figure 9. Dye Dispersion Curve, Republic Tertiary Pond.

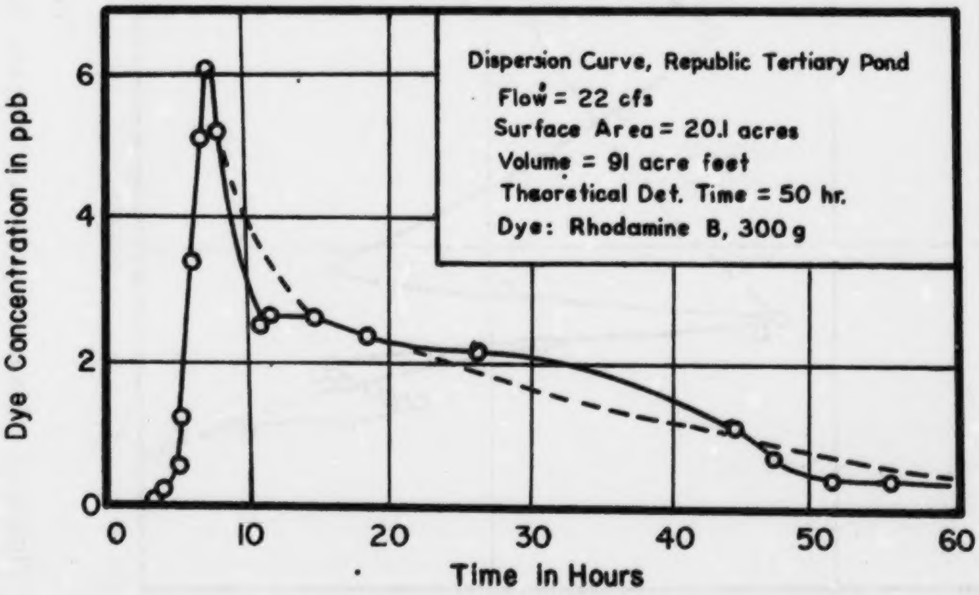

Figure 10. Dye Dispersion Curve, Republic Tertiary Pond.

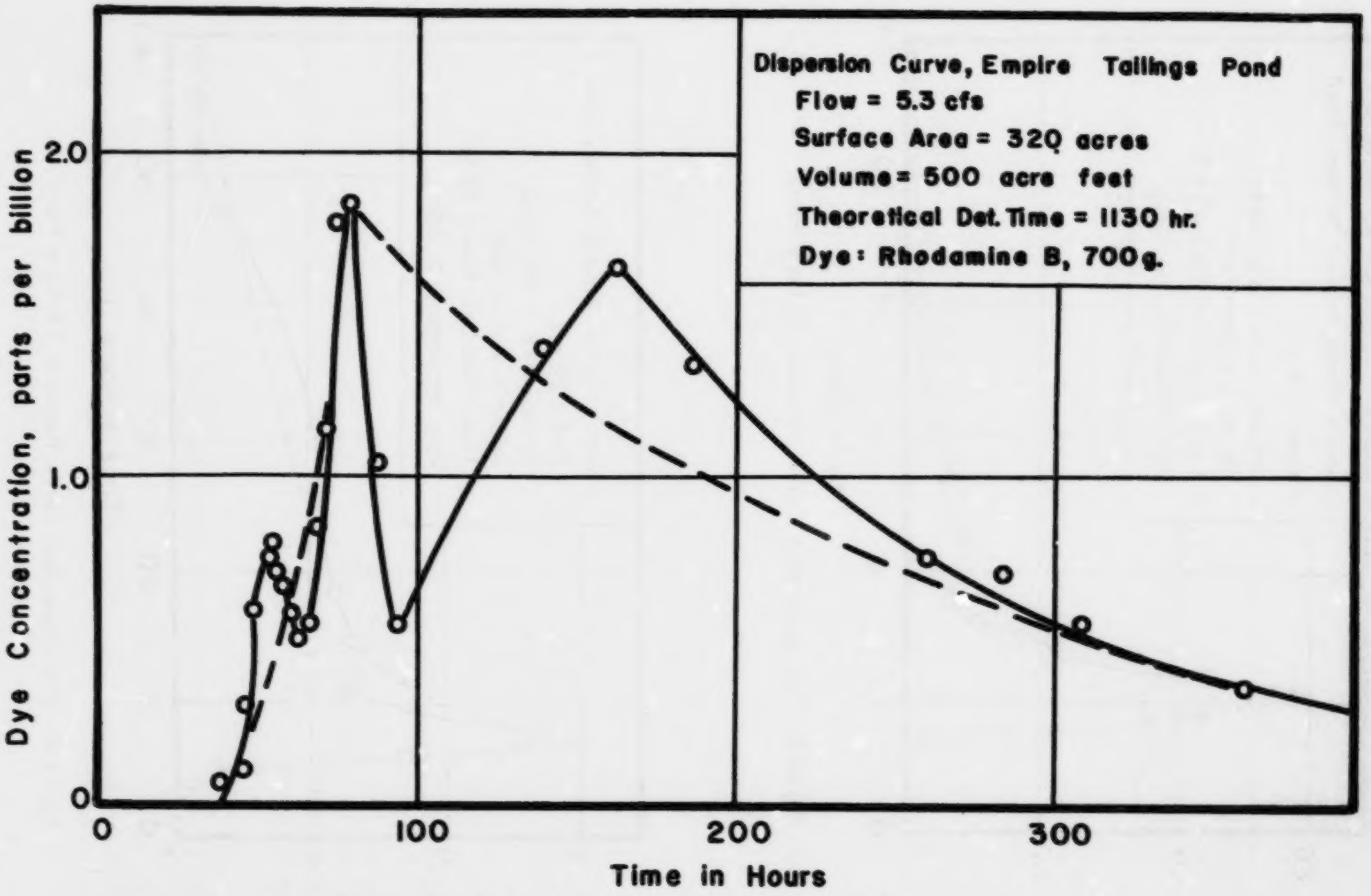

Figure 11. Dispersion Curve, Empire Clarification Basin.

Table 5 . Comparison of Hydraulic Characteristics of Tailings Clarification Basins

	Flow Rate, cfs	Initial Time Ratio	Modal Time Ratio	Dispersion Index
Republic				
Pond 3 (Northern Portion)	36	0.11	0.29	5.9
Tertiary Pond	28	0.056	0.11	4.1
Tertiary Pond	22	0.072	0.14	6.7
Empire Pond	5.3	0.033	0.071	5.2
Theoretical				
Plug Flow	—	1	1	1
Ideal Mixing	—	0	0	23.6
Representative Values for Rectangular Basins (5)	—	—	0.2-0.7	3-8

these observations seem reasonable. The calculated values for the dispersion index compare well with representative values for rectangular basins (5). Thus, it seems that the relatively poor hydraulic characteristics observed are caused primarily by short circuiting and stagnant regions.

It is obvious that improvements in plan configuration and inlet positions could produce hydraulic characteristics more closely approximating plug flow. However, in general, plan configuration and inlet position are largely dictated by topographical features. It follows, then, that under conditions producing poor hydraulic characteristics, only a fraction of the basin volume should be considered as effective.

Comparison of Predicted and Actual Effluent Concentrations

The method by which the settling column and dye dispersion data were combined to predict the effluent concentration for a non-ideal basin is explained by Baillod and Christenson (26) and outlined in Appendix D.

As stated earlier, data on the performance of the Republic Mine clarification basins were gathered over a one year period. These data included suspended solids concentrations,dissolved solids concentrations and flows at various points within the system and are summerized in Figures A.2, A.4 and A.5 of Appendix A and Table 7 on page 46. These data were used to compare the actual performance of the clarification basins with that predicted from the settling column analyses. The field performance was compared both with that predicted for an ideal basin and that predicted from the residence time distribution for a real flow basin. This was done to estimate the correction or scale-up factors involved in using settling column data to design large clarification basins.

Ideally, the comparisons between predicted and actual performance should be made for periods of steady-state operation during which the flow was equal to that existing at the time of the dispersion test. In addition, the other field conditions (e.g. influent suspended solids, dissolved solids, temperature) should correspond to those employed for the column settling test. However, with the exception of the periods during which the dispersion tests were conducted, true steady state conditions seldom existed for periods longer than a few hours. Thus, periods of approximately steady conditions were used for comparison. The column test conditions were chosen to reflect the conditions existing during these periods. Linear interpolation was employed to predict performance for field temperatures falling between two column tests.

Comparisons were made for the northern portion of Republic Pond 3 for 7 periods during which the flow rate was approximately equal to that existing during the dispersion test. Table 6 summarizes the actual and predicted effluent concentrations. The concentrations predicted by considering the residence time distribution agreed reasonably well with the actual concentrations. The concentrations predicted for an ideal plug flow basin with a volume equal to that of the actual basin were considerable less than the actual concentrations. The correction factors listed in column 6 represent factors by which the volume of the actual basin would have to be multiplied to produce an effluent concentration equivalent to that of the ideal basin. The reciprocal of this factor could be loosely interpreted as an effective fraction of the actual basin volume. In designing a basin based upon column settling data, the detention time required for an ideal basin to attain the specified concentration should be multiplied by this correction factor.

Figure 12 shows a correlation between the observed and predicted effluent concentrations. It is evident that the residence time distribution analysis was fairly successful in predicting the actual concentration. However, since few data related to residence time distributions in large impoundments are presently available, it would be difficult to apply this analysis in design. Although the effluent concentrations predicted for the ideal basin were considerably less than

Table 6. Summary of Actual and Predicted Effluent Suspended Solids Concentrations for Northern Portion of Republic Pond 3 At a Return Flow Rate of Approximately 35 cfs

Period	Water Temp. °F	Actual Effl. Susp. Solids mg/L	Predicted Effl. Susp. Solids mg/L		Corr. Factor For Ideal Basin
			Considering Residence Time Distribution	Ideal Basin	
(1)	(2)	(3)	(4)	(5)	(6)
Aug. 3-16	72	280	329	146	1.92
Sept. 21-28	55	320	430	191	1.68
Nov. 3-8	43	500	550	244	2.05
Feb. 23-27	32	720	741	328	2.20
Mar. 19-23	32	760	565	315	2.41
May 20-26	52	520	455	202	2.57
Sept. 1-14*	65	245	252	157	1.56

*Period of Dispersion Test

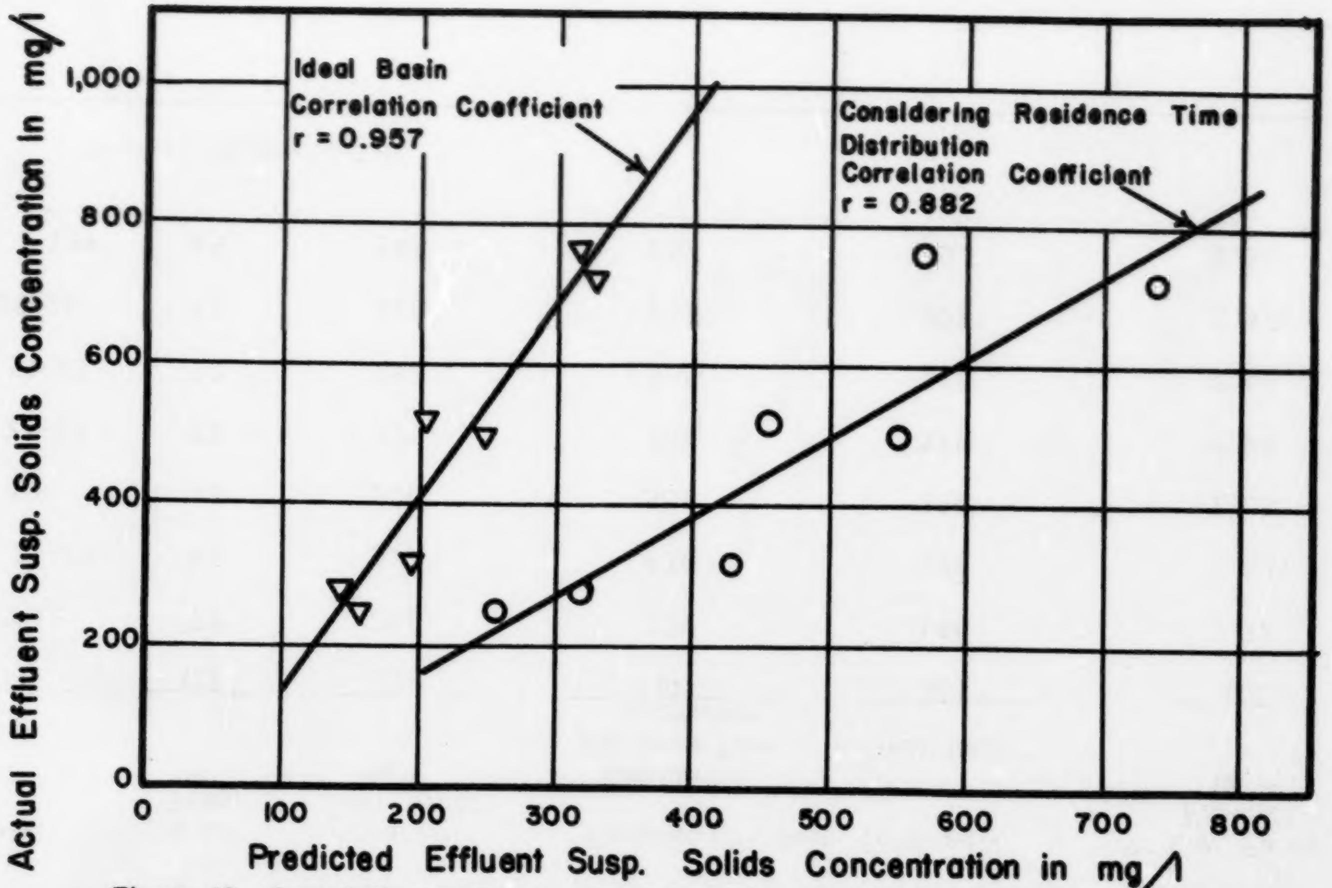

Figure 12. Correlation Between Actual and Predicted Effluent Suspended Solids Concentration.

those attained in the actual basin, a reasonable correlation existed. The correction factors for the ideal basin analysis ranged from 1.56 to 2.57 with the higher values occurring during the initial melt conditions. Thus, for basin performance under critical melt conditions, the larger factors would be appropriate. These factors would apply only to basins similar to Pond 3 at Republic. For different basin geometries, these factors would be expected to change.

The modal time ratios shown in Table 5 are indicative of the dead or stagnant regions within a basin. The ratio of 0.29 for Republic Pond 3 is significantly greater than the ratios for the other ponds, indicating that the effective fraction of the Pond 3 volume was greater than the effective volume fractions of the other ponds. If, as a first approximation, the ideal basin correction factors are assumed to be inversely proportional to the modal time ratios, rough estimates of ideal basin correction factors applicable to flocculent settling in the Republic Tertiary Pond and Empire Pond can be formulated. Taking a correction factor of 2.5 as corresponding to a modal time ratio of 0.29 yields correction factors of 6 and 10 for the Tertiary Pond and Empire Pond respectively.

In the case of the Republic Tertiary Pond a comparison of Figures 9 and 10 had shown the flow pattern to be unstable. Thus, for the Tertiary Pond, performance comparisons were made only for the periods during which the dispersion tests were conducted. At a flow rate of 22 cfs the actual effluent concentration was 18 mg/l. Under similar conditions, the settling column data predicted an effluent concentration of 3 mg/l for an ideal basin and 20 mg/l considering the residence time distribution. However, at a flow rate of 28 cfs, the actual concentration was 50 mg/l while the predicted concentration was 4 mg/l for both the ideal and non-ideal flow basins. The unstable flow pattern hampered the predictions in this case.

Application to the Design and Management of Tailings Storage and Clarification Basins

Topographical Considerations

In the design of a tailings storage and clarification system, the general configuration of the basins is influenced by the natural topography of the area. From an economic viewpoint, the most desirable site would be one which requires a minimal amount of dike construction, is situated in an area of low land value, and minimizes the operating expense.

Taken together, these three site attributes will minimize the cost per ton of tailings stored. According to these criteria, a natural lake is an extremely desirable site since dike construction can be avoided and, in certain instances, the lake bottom, upon which the tailings are stored, may be "free" government land. Thus, in the Lake Superior region, natural lakes have often been employed for tailings storage.

A second type of desirable site from a topographical viewpoint would consist of natural valley with steep walls. With this topography, the tailings deposit could be retained by a single dike across the lower end of the valley. In practice, however, because of land costs and transportation expense, the most economical site may have topographical features which necessitate more dike construction than the sites mentioned above. The topography of the Republic tailings site, for example, is characterized by several small hills or knolls surrounded by relatively flat land. In this situation, the amount of dike construction can be minimized by taking advantage of the higher areas of natural terrain. Thus, to a certain extent, the plan configuration of the impoundment and clarification system has been influenced by the site topography. The Empire tailings site is located in a valley forming the headwaters of Green Creek. The principal dike, therefore, has been constructed across the lower end of the valley and forms the southeast boundary of the tailings impoundment.

It is evident that, in any given location, the plan and profile configuration of a tailings impoundment and clarification system will, to a great extent, be governed by the site topography.

Hydrologic Considerations

From the preceeding discussion it can be seen that the impoundment and clarification sites most desirable as far as minimizing dike construction is concerned generally result in a certain area of natural watershed being tributary to the impoundment and clarification system. This tributary watershed, in itself, does not affect the volume available for tailings storage. However, the runoff from this tributary watershed will affect the volume required for clarification. Table 3 indicated that, at the Empire site, the excess volume (above and beyond the average) released during the spring amounted to 25.8 in. spread over 726 acres, or 1,560 acre feet. Nearly 50% of this resulted from snow accumulated on the 1,315 acres of tributary watershed. The excess flow at the Republic system, with no tributary watershed amounted to only 15.5 inches spread over 938 acres, or 1,210 acre feet. Thus, as far as the design and operation of the clarification system is concerned, it is preferable to avoid large areas of tributary watershed.

The manner in which the tailings deposit is formed can also influence the excess flows released during the spring melt and in turn affect the volume required for clarification. The overland sheet flow at Republic caused a build-up of tailings ice and ground storage amounting to 13.7 inches over 938 acres, or 1,070 acre feet. At Empire the overland sheet flow covered a smaller area and most of the solids were deposited into open water. This resulted in a maximal tailings ice and ground storage accumulation of 10.1 inches over 726 acres, or 610 acre feet. Thus, with all other considerations being equal, it is preferable to avoid the occurrence of overland sheet flow during the cold months. However, it should be pointed out that the overland sheet flow is effective in increasing the elevation of the tailings deposit. Therefore, in many cases this mode of operation may be necessary.

In the rational design of a tailings clarification system, it is necessary to determine a design flow for clarification. (i.e. the maximum effluent flow rate which the clarification system can pass while still maintaining a satisfactory effluent quality.) This will normally be less than the flow rate employed for hydraulic design. If provision for storage is to be made in the clarification basin, the maximal flow rates into the basin will be greater than the maximal effluent flow rates. In this case, it is logical to base the clarification design on the basin effluent flow rate. The design storage will then be determined by both the influent and effluent flow rates. In addition, an averaging time must be associated with the clarification design flow rate. In waste treatment practice, the averaging times associated with process design flows are generally approximated by the detention time of the unit to be designed. For tailings clarification basins, this averaging period could vary from 1 to 15 days depending on the size of the basin and the flow rate.

It is evident from the hydrographs given in Figures A.1 and A.2, (Appendix A) that, in the Lake Superior Region, the critical flow conditions will occur during the spring melt. The designer is faced with constructing an approximate hydrograph for the expected inflow to the tailings clarification basin during the critical melt period. If storage is to be provided, the instantaneous peak inflow rates are not overly important. Thus, this approximate hydrograph might be based upon 10 day mean inflow rates. The hydrograph would be constructed by superimposing the excess flow released during the melt on the base flow (plant inflow less uniform storage less seepage) as illustrated in Figure 13. The total area above the base flow represents the volume of excess water released during the melt period and would consist of snow, tailings, ice, precipitation during the melt period and possible ground water storage. The volumes of snow (as water equivalent) and precipitation to be expected for any given recurrence interval can be easily estimated by a frequency analysis. The volumes of tailings ice and ground water storage are more difficult to estimate. The values listed in Table 3 might be judiciously employed as guides in estimating tailings ice and ground water for systems closely related to the Republic and Empire systems.

The exact shape of the inflow hydrograph during the melt period is not critical. Of more importance is the base width, which is difficult to accurately estimate. In the case of a basin dominated by a tributary watershed it may be possible to gather flow data before construction of the tailings basin begins. Alternatively, values similar to those observed in this study might be cautiously applied to closely similar systems.

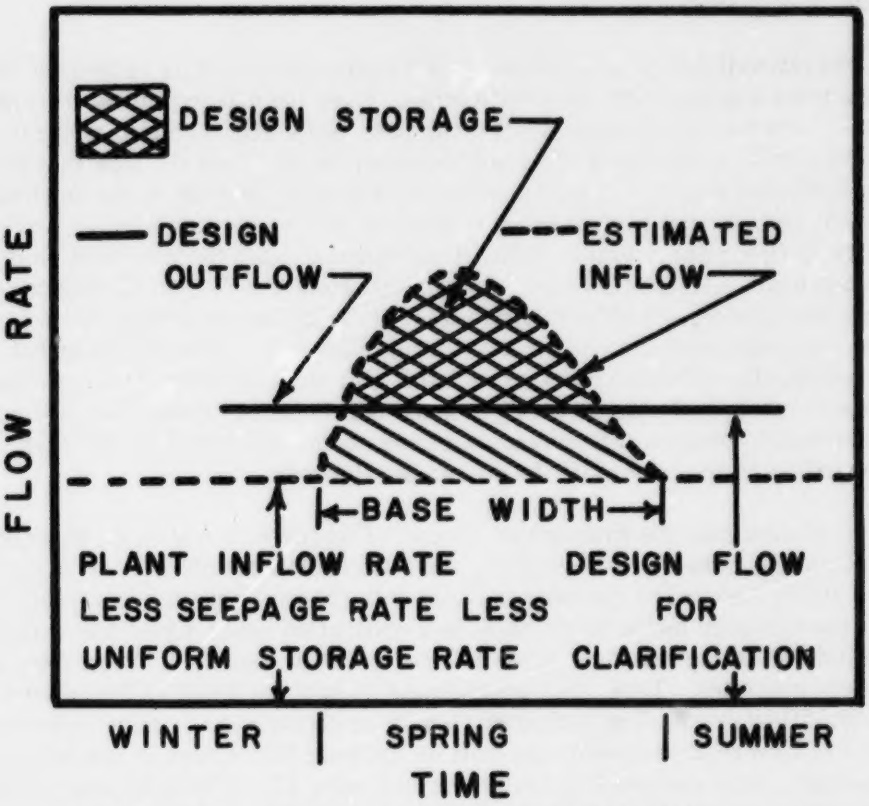
DESIGN STORAGE
DESIGN OUTFLOW
ESTIMATED INFLOW
FLOW RATE
BASE WIDTH
PLANT INFLOW RATE LESS SEEPAGE RATE LESS UNIFORM STORAGE RATE
DESIGN FLOW FOR CLARIFICATION
WINTER
SPRING
SUMMER
TIME

Figure 13. Conceptual Sketch of Hydrograph during Spring Melt Conditions.

Once the base width is established and the total volume of excess water to be released during the spring is estimated, it is possible to sketch the estimated inflow hydrograph. The design storage required in order that any specified design outflow not be exceeded is given by the doubled shade area illustrated in Figure 13. It can be seen from the figure, that, for a given situation, increasing the design flow for clarification decreases the design storage. At the same time, however, increasing the clarification design flow increases the required surface area of the clarification basin. The design storage would normally be provided by raising the surface elevation of the clarification basin. Thus, greater storages would necessitate higher dikes whereas a greater clarification area would require more land area and possibly longer dikes. The optimal design outflow is that flow which minimizes the overall cost while still yielding an effluent of the desired quality.

Based on data collected during the study period, the base flows for the Republic and Empire tailings systems can be estimated as follows:

Base Flow = Plant Inflow Rate - Seepage Rate - Uniform Storage Rate

At Republic,

Base Flow = 58 cfs - 8.9 cfs - (+ 1.30 cfs - 1.41 cfs)

Base Flow 49.2 cfs

The base flow at Empire can be expressed as:

Base Flow = 9.9 cfs - 7.75 cfs - (- 1.31 cfs) = 3.46 cfs

Balance calculations indicate that, in the absence of storage, outflow rates higher than these base flows would have occurred over a period of 45 days at Republic and 40 days at Empire. As pointed out earlier, however, the high flow period began earlier at Republic.

In certain instances, the designer may be constrained by maximum (and perhaps minimum) basin outflow rates specified by regulatory agencies. The minimum design flow for clarification would be the average annual outflow. At this outflow rate, storage would be maximized and the accumulated storage would be gradually released during dry periods.

In the operational management of a tailings impoundment and clarification system, it is essential that attention be given to the hydrology of the basin, particularly during snowmelt conditions. Prior to, and during melt conditions, the operator should be aware of the water equivalent of the snow pack and tailings ice resting on the tributary basin. However, not even the most refined operational management can result in satisfactory operation of a system in which adequate storage volume and clarification area are not available.

Water Reuse:

From consideration of the overall water balances, it is obvious that reuse of the clarified process water can reduce the quantity of water discharged from the tailings basins and thereby lessen the impact on receiving waters. Figure 2 indicates that at the Republic Mine an average of about 63% of the process water is reclaimed from the tailings basin. Likewise, Figure 4 shows that, at the Empire Mine, 96% of the process water is reclaimed from the tailings thickener.

In general, reuse of spent process water may be limited by the following factors:

1. A certain quantity of water must remain in the tailings stream to effect fluid transport of the waste solids.
2. The presence of dissolved and suspended material may interfere with concentration processes. It is known for example, that di and tri valent cations can precipitate certain flotation reagents thereby increasing process chemical requirements.

At both sites, the dissolved solids in the mill effluents were considerably higher than that in the fresh make-up water. Approximate ranges of dissolved solids concentrations observed at various points in the tailings systems are indicated in Table 7. The concentrations measured in the pond effluents tended to vary seasonally being high in the winter and summer and low in the spring because of the dilution from melting ice.

Table 7. Approximate Ranges of Dissolved Solids Concentrations (ppm) Observed At Various Points in Tailings Systems

Empire			Republic		
Make-Up Water	Mill Effluent	Pond Effluent	Make-Up Water	Mill Effluent	Pond Effluent
75	300-380	150-240	60	110-150	90-130

It is evident that certain materials are solubilized during the concentration process and precipitated in the impoundment system.

The problem of water reuse does not lend itself to a simple analysis, as one is immediately faced with cost trade-offs between treatment of pond discharge, concentration process cost, and treatment of the reuse water. Assuming that a concentrating plant is faced with a set of effluent standards, techniques of systems analysis (5) could be applied to determine the optimal reuse strategy. However, before this can be done, some basic information related to reuse would have to be established. This necessary information would include:

1. Water quality criteria for reuse. It would be necessary to develop information on allowable concentrations of various substances in water to be reused in various concentration schemes applied to various ore types. Information on incremental concentrate production costs associated with higher concentrations of these substances would also be required.

2. An applicable methodology to predict any build-up of important substances which might occur within a reuse system.

Further research on this topic is obviously needed since this information is not presently available.

SECTION V
TREATMENT OF TAILINGS BASIN OVERFLOWS

The primary objective of this aspect of the research was the investigation of alternative treatment methods which might be applied both to remove particulate material from the basin discharge and to facilitate final disposal of the collected solids. A related, secondary objective, was to assess the feasibility of incorporating certain treatment methods into a treatment system for subsequent demonstration.

The research plan employed to achieve these objectives relied heavily on experimentation with the highly colored 5B outlet stream and tertiary pond at the Republic Mine. The Empire Mine tailings pond overflow contained very low suspended solids concentrations (on the order of 10 ppm) and thus did not lend itself to this aspect of the research.

Investigation of Alternative Coagulants

It is difficult to conceive a process which could effectively replace chemical coagulation and sedimentation in the treatment of tailings basin overflows. Thus, some effort was devoted to the development of refinements in the coagulation flocculation process. The basic experiemtnal procedure employed here consisted of the well-known jar test. A detailed description of the methodology is given in Appendix E.

Initial Screening of Various Coagulants

Alum and thirteen polymer coagulants or coagulant aids were initially selected for jar testing. The polymers consisted of: 5 anionic coagulants, 1 anionic coagulant aid, 5 cationic coagulants (one of which was also tested as a coagulant aid) 1 non-ionic coagulant, and 1 non-ionic coagulant aid as shown in Table 8. The term "coagulant aid" denotes an agent employed in conjunction with another coagulant. In this investigation, coagulant aids were evaluated in conjunction with alum. Several other polymers were excluded from testing for one of the following reasons: possible toxicity; extreme difficulties involved in preparing standard polymer solutions; or, in the case of one company, no information was received regarding type, recommended usage, or price of the polymer samples.

The initial screening jar tests were conducted under the following conditions;

Temperature = 13°C
Initial pH = 7.5
Initial Turbidity = 1,400 Jackson Units

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Table 8 . Coagulants Subjected to Initial Screening Tests

Type	Manufacturer	Manufacturer's Designation	Given Cost $/lb.
Alum			$0.03
Anionic Polymer Coagulant	Calgon	M-560	$1.38
" " "	"	M-570	$1.38
" " "	"	M-580	$1.38
" " "	"	M-590	$1.38
" " "	Nalco	D-2332	$0.95
Anionic Polymer Coagulant Aid	Dow	A-22	$1.15
Cotionic Polymer Coagulant	Calgon	M-500	$0.43
" " "	"	M-510	$0.44
" " "	"	M-520	$2.00
" " "	Nalco	Na-603*	$0.241
" " "	"	Na-607	$0.30
Non-Ionic Polymer Coagulant	Calgon	M-550	$1.38
Non-Ionic Polymer Coagulant Aid	Dow	N-11	$0.95

*Also tested as a coagulant aid.

To facilitate subsequent comparisons between coagulants, a system of coding the various polymers was developed. This method is explained in Table 9 .

Table 9 . Polymer Coding System.

1st Letter	2nd Letter	Manufacturer's Designation
A = anionic polymer	C = Calgon	Refers to a company's
C = cationic polymer	D = Dow	own polymer number.
N = non-ionic polymer	N = Nalco	

For example, polymer coagulating agent ACM-560 is an anionic polymer manufactured by the Calgon Corporation, with a model number of M-560.

Results of Initial Screening Tests

Anionic Polymer Coagulants. Polymers ACM-560 to ACM-590 (model numbers pertain to relative values of polymer molecular weights) exhibited certain identifiable characteristics, summarized below.

(1) All polymers exhibited rapid, massive floc formation following their addition to the samples being jar tested. The speed of floc formation, and the size of the irregularly-shaped floc clumps increased with polymer dosage and polymer molecular weight.

(2) Floc particles formed by higher dosages of all polymers exhibited significant deposition during slow mixing operations. Higher dosages of polymer ACM-590 formed extremely massive, irregularly-shaped, floc particles which settled out during rapid mixing operations.

(3) Increasing dosages and molecular weight of all polymers resulted in increasing supernatant color remaining after coagulation-flocculation-quiescent settling. This was not related to levels of residual supernatant turbidity, however, as turbidity removals increased (or decreased slightly) with increasing polymer dosages.

(4) Coagulation-flocculation with the polymers tested did not significantly reduce sample pH or total alkalinity.

Anionic polymer coagulant AND-2332 exhibited the poorest floc formation and settling characteristics of all the anionic polymer coagulants tested.

Figure 14 depicts turbidity removal efficiency as a function of increasing polymer dosage. It is observed that at dosages above 6 ppm, the lowest molecular-weight polymer, ACM-560, gave the highest removals. Increasing dosages (beyond 10 ppm) of the higher modecular-weight anionic polymers generally resulted in decreased removal efficiencies.

Anionic Polymer Coagulant Aid. An additional variable, alum concentration, is introduced when comparing efficiencies of coagulant aids. It proved convenient to make these comparisons at equal levels of chemical cost per million gallons treated. Thus, the turbidity removal efficiencies of various, equal cost, combinations of alum and coagulant aid were compared. These comparisons were made over the cost range of 0 to $25 per million gallons. The cost of $25 per million gallons was equal to the chemical cost at the optimal alum dosage of 100 ppm. Cost data for the various coagulants are given in Table 8 .

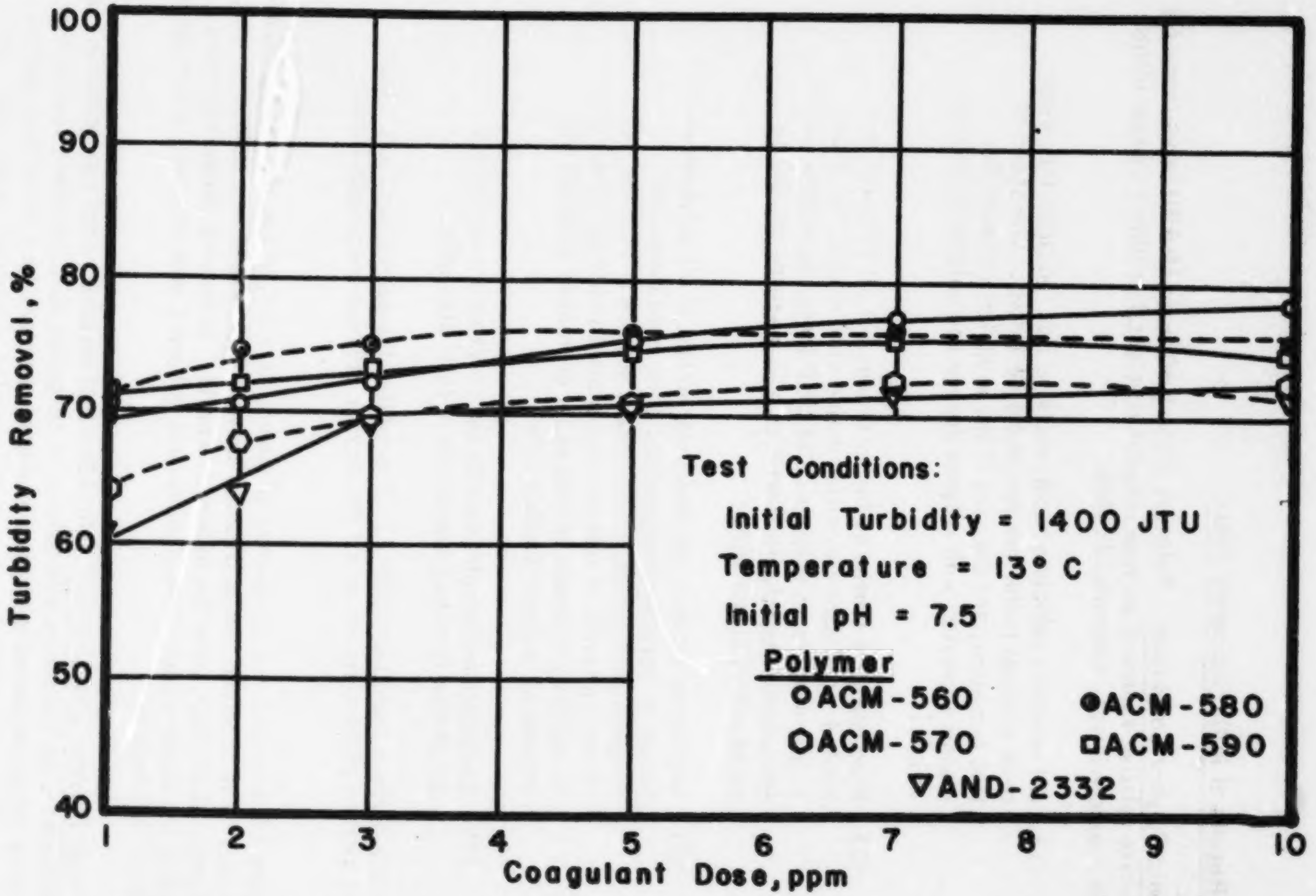


Figure 14. Comparison of Turbidity Removals for Several Anionic Polymer Coagulants.

The turbidity removal efficiency of anionic polymer coagulant aid ADA-22 was dependent on the amount of alum added to the system, as can be seen in Figure 15. At a constant alum dosage, of 10 ppm, no amount of the coagulant aid added to the system resulted in turbidity removals comparable to those obtained with increasing dosages of alum alone. Alum dosages of 30 to 50 ppm, together with increasing dosages of coagulant aid, resulted in turbidity removal values higher than identical-cost alum dosages. No further improvement in supernate was obtained at the highest coagulant aid dosages.

Cationic Polymer Coagulants. Cationic polymer coagulants CCM-500 to CCM-520 (model numbers pertain to relative values of polymer molecular weights) exhibited certain common characteristics, relative to polymer dosage and molecular weight; these are summarized below.

(1) As the polymer molecular weight increased, the turbidity removal efficiency obtained with low polymer dosages increased.

(2) The speed of floc formation, and the size of floc particles formed during a jar test, increased with increasing polymer molecular weight.

(3) The density of floc particles decreased with increasing polymer molecular weight.

(4) As molecular weight of the cationic polymer coagulants increased, amounts of standing floc (surface scum) formed during jar testing decreased.

(5) All 'CCM' series cationic polymer coagulants were effective in removing fine turbidity-causing particles, with CCM-510 being most effective.

(6) All polymers exhibited extremely rapid floc particle settling, leaving a clear supernatant within 2-3 minutes of quiescent settling.

Polymer coagulants CNNa-603 and 607 were significantly less effective than coagulants CCM-500 to CCM-520. The higher molecular-weight polymer, CNNa-607, exhibited the poorest floc formation and floc particle settling characteristics of all cationic polymer coagulants evaluated.

Figure 16 depicts per cent turbidity removal as a function of increasing cationic polymer coagulant dosages. Decreased removal efficiency (and subsequent increases in sample supernatant turbidity) occurred only with higher dosages of

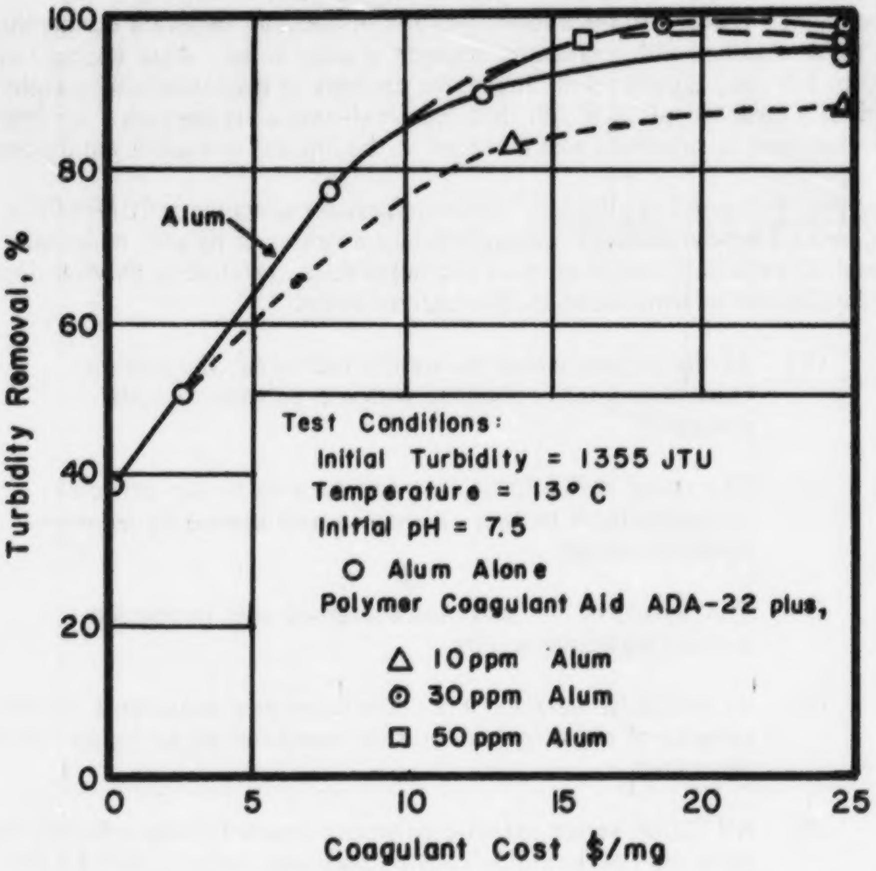


Figure 15. Turbidity Removal of Various, Equal Cost Combinations of Alum and Anionic Polymer Coagulant Aid.

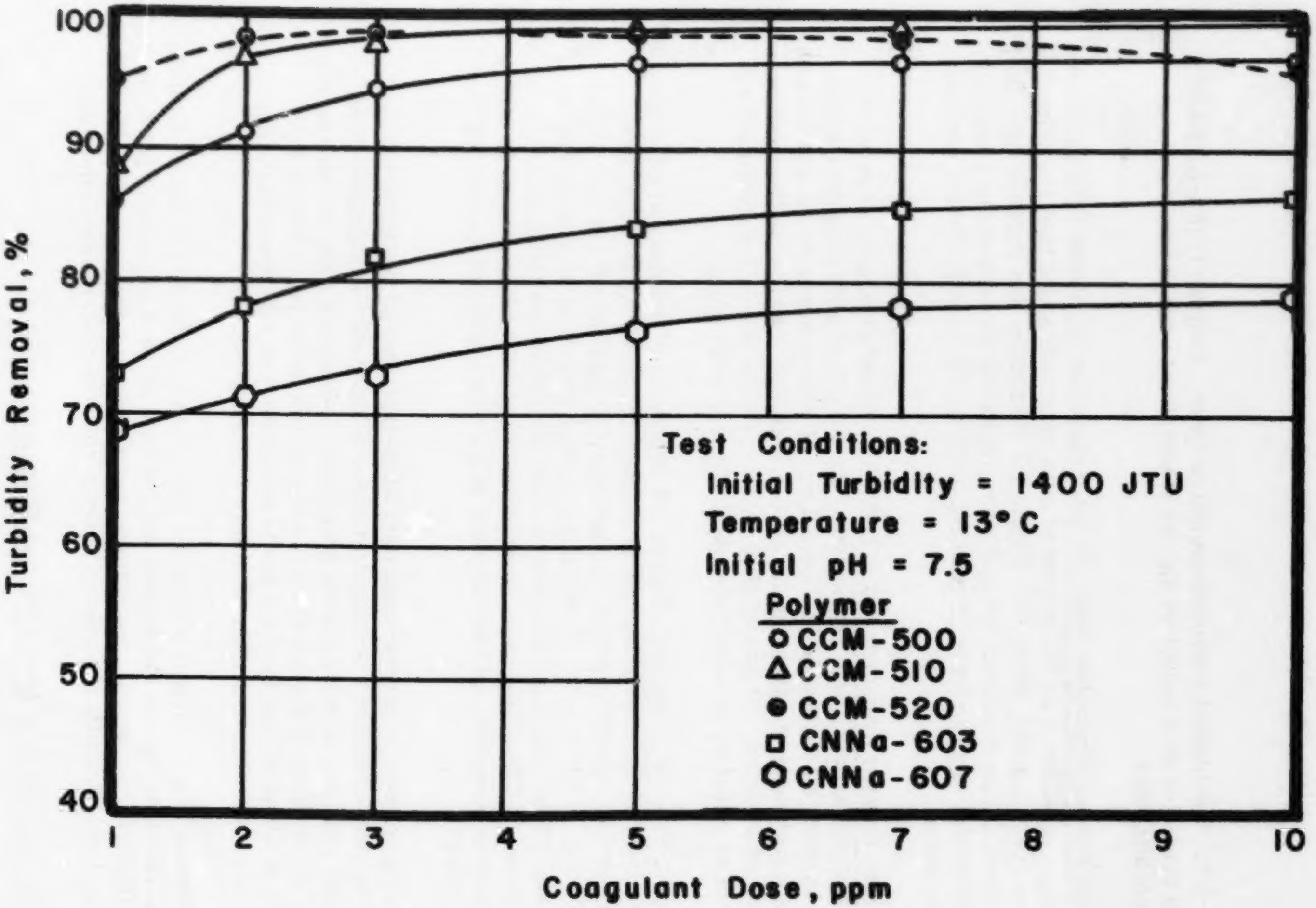


Figure 16. Comparison of Turbidity Removals for Several Cationic Polymer Coagulants.

CCM-520, the highest molecular-weight polymer. Polymer CNNa-603 was further evaluated as a coagulant aid, as it was being used in this manner at the Republic Mine.

Cationic Polymer Coagulant Aid. Polymer coagulant aid CNNa-603 exhibited a stronger dependence on the amount of alum coagulant added than the anionic polymer coagulant aid, ADA-22. Figure 17 illustrates this dependency. Only at a constant alum dosage of 50 ppm were increasing amounts of the coagulant aid successful in removing more per cent turbidity than identical-cost dosages of alum alone.

Non-Ionic Polymer Coagulant Aid. Turbidity removal efficiency of polymer coagulant aid NDN-11 was dependent on the amount of alum coagulant added to the system, as illustrated in Figure 18. At alum dosages of 30 and 50 ppm, increasing amounts of coagulant aid greatly improved coagulation-flocculation process efficiencies; 10 ppm of alum plus increasing amounts of coagulant aid were not as effective as identical-cost dosages of alum alone.

Non-Ionic Polymer Coagulant. Figure 19 depicts turbidity removal efficiency as a function of increasing dosages of polymer coagulant NCM-550, and illustrates three distinct zones of coagulation-flocculation efficiency: (1) an initial zone, in which removal efficiency greatly increased as more coagulant was added to the samples; (2) a narrow zone of optimum turbidity removal efficiency; and (3) a final zone, in which turbidity removal efficiency greatly decreased as more than optimum amounts of polymer coagulant were added to the samples.

Increasing dosages of polymer coagulant NCM-550 formed the largest floc of any polymer coagulants or coagulant aids. Floc formed immediately after coagulant addition in rapid mixing operations. Subsequent slow mixing operations caused the breakup of these massive, clumped floc particles. Substantial amounts of standing floc (surface scum) were formed at all coagulant dosages. No effect on sample pH or total alkalinity was observed.

For purposes of comparison, Figure 19 also shows, coagulant dosage - turbidity removal relationships for the anionic and cationic polymer coagulants judged to be most effective. In general, the cationic polymer coagulants were most effective in removing turbidity while the anionic polymer coagulants were least effective.

Extended Evaluation of Most Effective Coagulants

Figure 19 shows that, of the polymer coagulants, CCM-510 was the most effective. Likewise, an examination of Figures 15, 17, and 18 indicates that, of the coagulant aids, NDN-11, was somewhat more effective than the others, in augmenting the removal by alum. Consequently, these two polymers, along

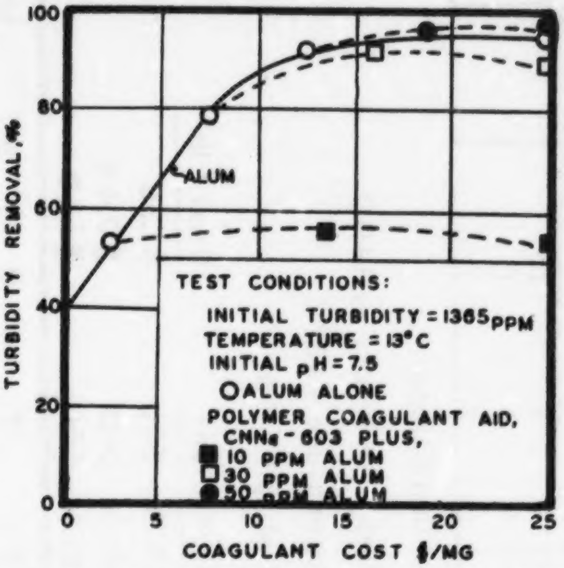


Figure 17. Turbidity Removal of Various, Equal Cost Combinations of Alum and Cationic Polymer Coagulant Aid.

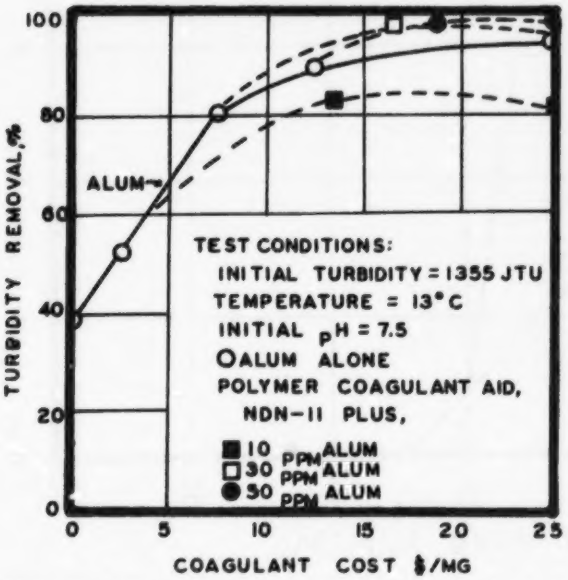


Figure 18. Turbidity Removal of Various, Equal Cost Combinations of Alum and Cationic Polymer Coagulant Aid.

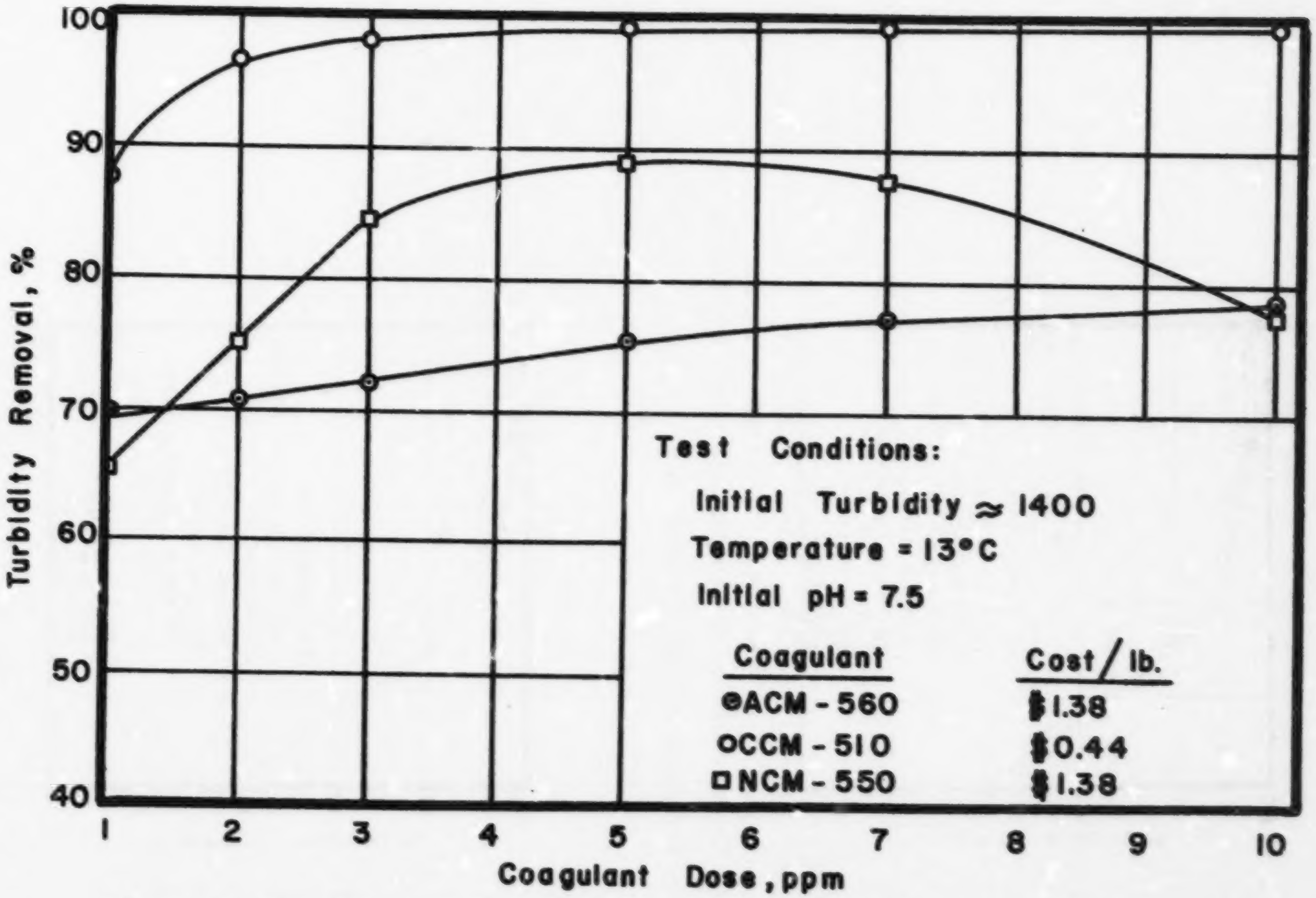


Figure 19. Comparison of Turbidity Removals for Most Effective Anionic, Cationic and Non Ionic Polymer Coagulants.

with alum, were selected for further testing. The purpose of these detailed evaluations was to determine the optimal coagulants for various temperature and turbidity levels. Thus, the extended evaluations consisted of turbidity removal comparisons at various equal cost combinations of coagulant and coagulant aid, and at various temperatures and initial turbidity levels. Initial turbidity levels of approximately 400, 700, 1000, and 1400 JTU were selected; 1400 JTU closely approximated the maximum yearly initial turbidity level recorded at the Republic Mine tailings basin overflow while 700 JTU was approximately equal to the average yearly initial JTU level. 1, 13, and 25° Centigrade represented the minimum, average and maximum overflow temperatures respectively. All jar tests were conducted at optimum slow mixing and initial pH conditions. Rapid mixing conditions were standard (see Appendix E).

Figure 20 shows turbidity remaining as a function of coagulant cost for each of the three coagulants. The effects of sample initial turbidity and temperature levels upon comparative turbidity reductions of the three coagulating agents are evident. No single coagulant or combination of coagulants was found to to be most economical in achieving a given supernatant turbidity under all conditions of initial turbidity and temperature. Chemical costs to achieve a specified turbidity varied with initial turbidity and temperature. However, for supernatant turbidities greater than about 60 JTU, Calgon M-510 was the most economical coagulant under all conditions of initial turbidity and temperature. Approximate chemical costs for Calgon M-510 required to attain a supernatant turbidity of 60 JTU ranged from $2.50/million gallons at an initial turbidity of 400 JTU and temperature of 25°C to $8.00/million gallons at an initial turbidity of 1400 JTU and temperature of 1°C. At low initial turbidities (400 JTU), this cost to achieve a supernatant turbidity of 60 JTU was virtually equal to the cost of alum plus coagulant aid Dow N-11 and was about 80% of the cost of alum alone. However, at high initial turbidities (1400 JTU), the cost of Calgon M-510 to achieve a supernatant turbidity of 60 JTU was roughly 50% of the cost of alum plus coagulant aid Dow N-11 and roughly 35% of the cost of alum alone.

Comparison of Sludge Volumes Formed by Various Coagulants

In an overall coagulation-flocculation-sedimentation treatment process, consideration of the thickening capacity of the sludge produced is, perhaps, just as important as the economic considerations involved in selecting a coagulant for clarification purposes. Therefore, observations were made to qualitatively compare the volumes of sludge generated by the various coagulants.

Upon completion of jar tests in which reasonably high turbidity removals were obtained, the samples were gently hand-mixed and allowed to settle in graduated Imhoff cones for 30 minutes. The volumes of sludge were routinely noted. Knowing the initial and final turbidities, it was possible, with the aid of the suspended

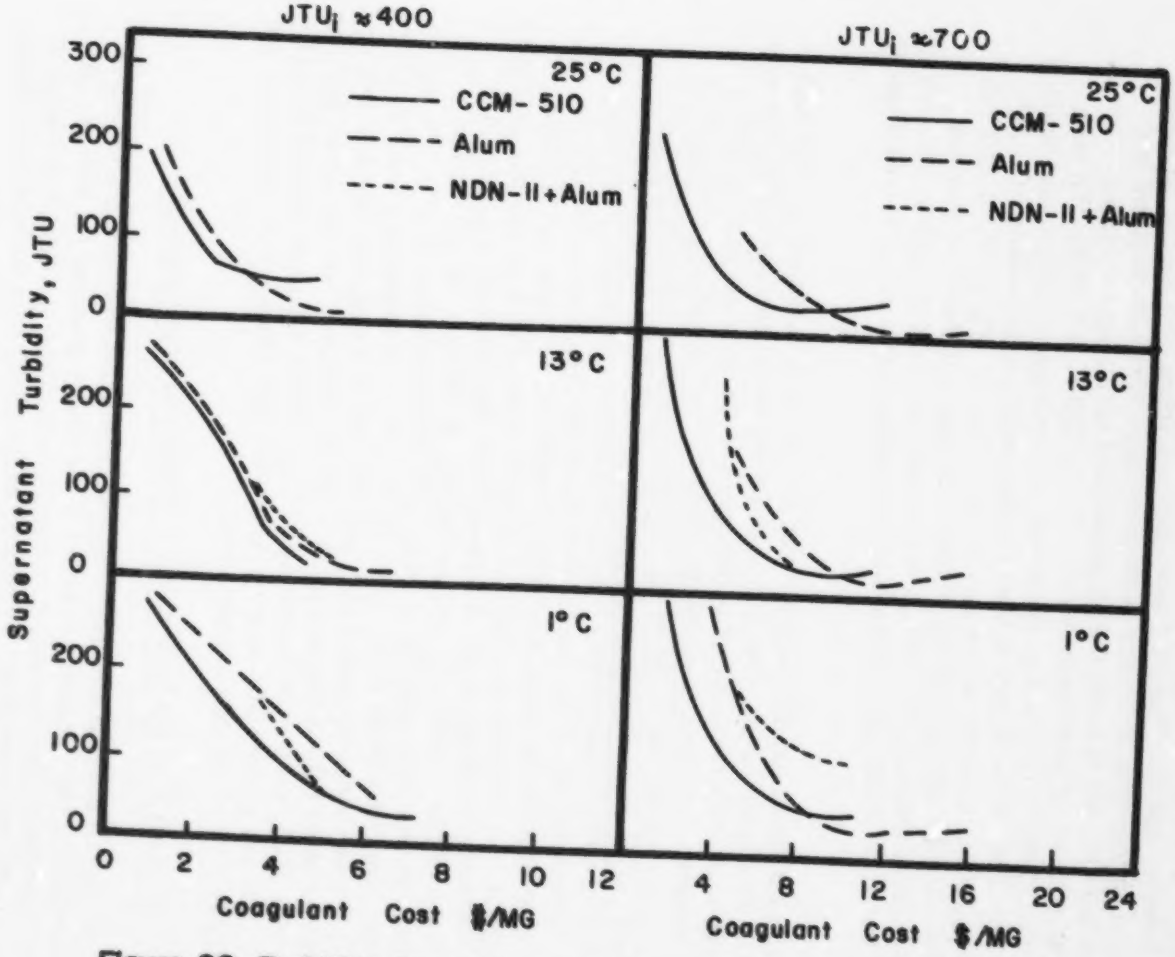


Figure 20. Turbidity Remaining as a Function of Coagulant Cost.

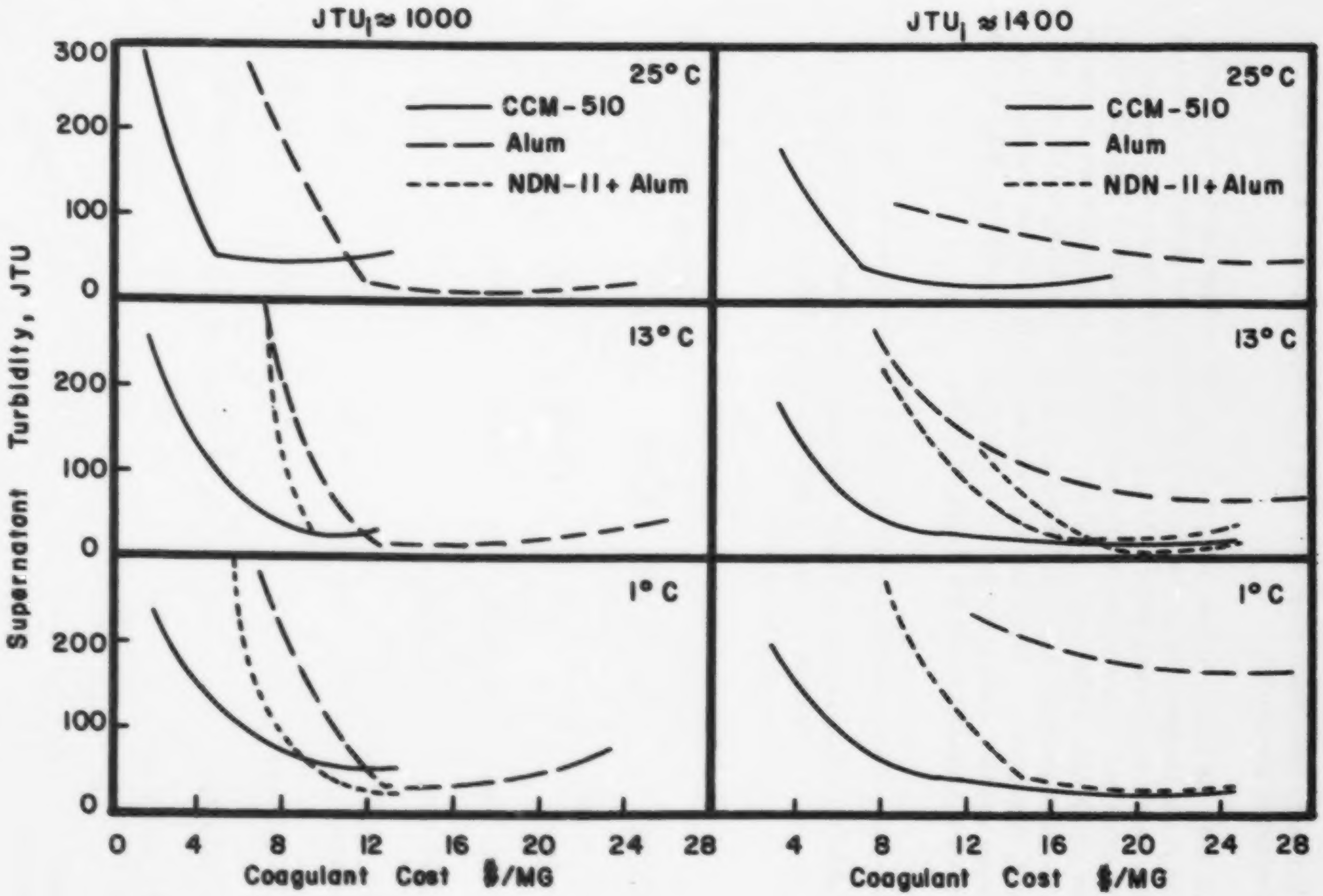


Figure 20. Continued

solids-turbidity correlation given in Figure E.1 (Appendix E), to estimate the solids content of the sludge.

Table 10 lists the sludge solids concentrations obtained for various coagulants. Data used in the constructing of this table were selected so that the supernatant turbidities were roughly comparable. Thus, comparisons can be made between coagulant costs and sludge volumes for equal initial turbidities. The CCM-510 coagulant which was, under most conditions, the most economical from a clarification viewpoint produced the largest sludge volume. The CNN_a-603, on the other hand, employed as a coagulant aid, was more expensive but produced the least sludge volume. During the period of this study, the polymer CNN_a-603 was employed as a coagulant aid at the Republic Mine.

Table 10. Comparison of Sludge Volumes Formed After 30 Minutes of Quiescent Settling in Imhoff Cones. Initial JTU = 1400. Temperature = 13°C.

Coagulant	Supernatant Turbidity, JTU after 30 min.	Coagulant Cost $/MG	ml Sludge/ liter	Approx. Sludge Solids Conc.%
NDN-11, 1 ppm + 30 ppm Alum	27	15.40	20	3.6%
CCM-510, 3 ppm	28	11.00	24	3.0%
Alum, 100 ppm	66	25.00	15	4.5%
ADA-22, 0.89 ppm 30 ppm Alum	38	16.00	17	4.2%
CCM-510, 3 ppm	28	11.00	24	3.0%
Alum, 100 ppm	66	25.00	15	4.5%
ADA-22,0.89 ppm 30 ppm Alum	38	16.00	17	4.2%
CNN_a-603, 3.1 ppm + 50 ppm Alum	47	18.70	10.5	6.4%

Characterization and Analysis of Republic Mine Coagulation-Flocculation System.

The coagulation-flocculation system at the Republic Mine consisted of:

(1) Coagulant addition.

(2) A weir, hydraulic jump and a relatively steep section of stream channel for rapid mixing (slope = 0.12 ft/ft for 50 ft).

(3) A relatively flat section of stream channel for slow mixing (slope = 0.0024 ft/ft for 800 ft).

In order to judge the effectiveness of this system, comparisons were made between the flocculation parameters attained in this system and those judged to be optimal based on laboratory jar tests. The mean velocity gradient, G, and the dimensionless product of G times residence time ·, GT, were the flocculation parameters employed (9).

Equation 9 developed for baffled channels (9) was used to estimate the mean velocity gradient, G, and GT for the rapid and slow mixing portions of the Republic Mine effluent channel.

$$G = \left(\frac{g \Delta h}{\nu T}\right)^{0.5} \qquad 9.$$

where:

G = mean velocity gradient fps/ft
g = accelleration of gravity, 32.2 ft/sec^2
Δh = drop in water surface, ft
ν = kinematic viscosity, ft^2/sec
T = mixing time, sec

The mixing time, T, was taken to be the mean residence time determined from a dye dispersion test at a flow rate of 28.3 cfs. The mixing time at other flow rates was estimated by assuming that channel roughness controlled the water depth and applying the Manning Equation (9) with channel depth equal to hydraulic radius.

Figures 21and 22 show the calculated values of the mean velocity gradient, G, and GT for the rapid and slow mixing portions of the effluent channel. Recommended values of these parameters for rapid mixing are (10):

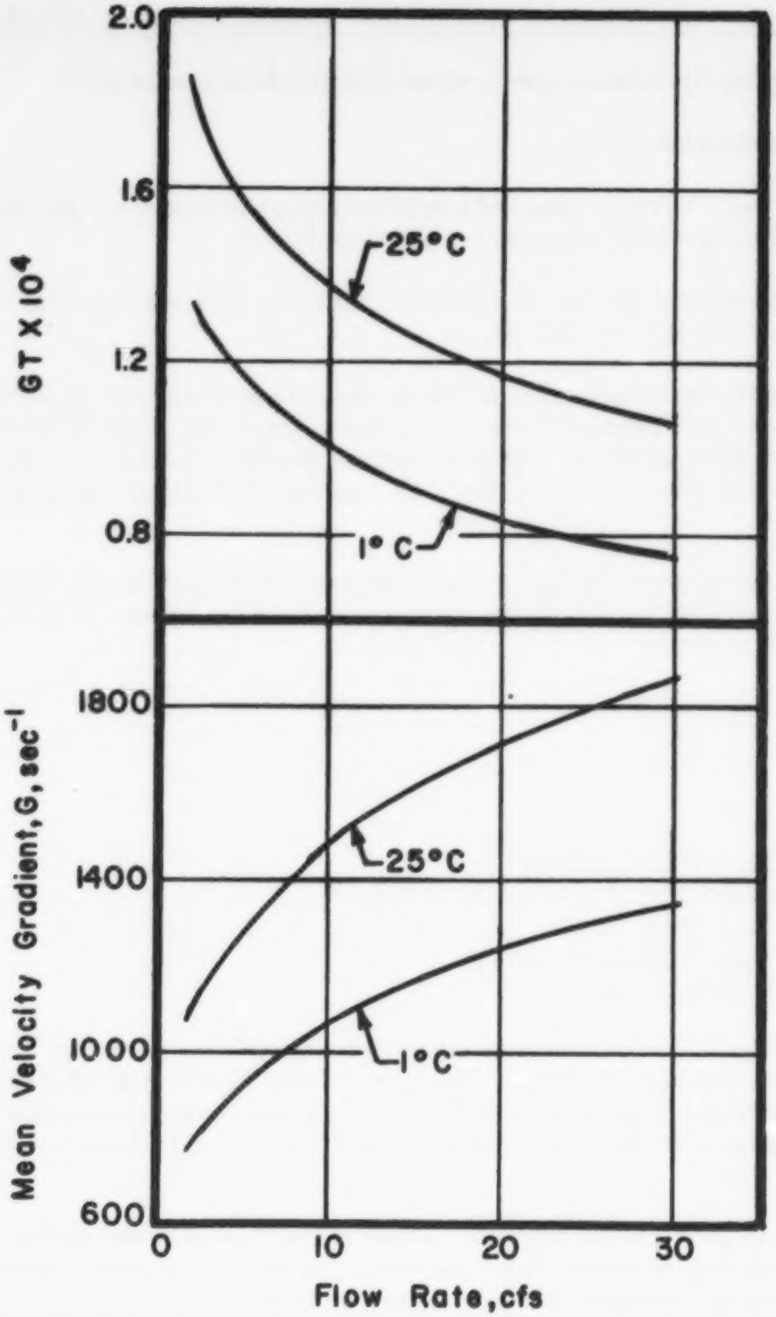


Figure 21. Calculated Values of G and GT as a Function of Flow Rate for the Rapid Mixing Portion of the Republic Effluent Channel.

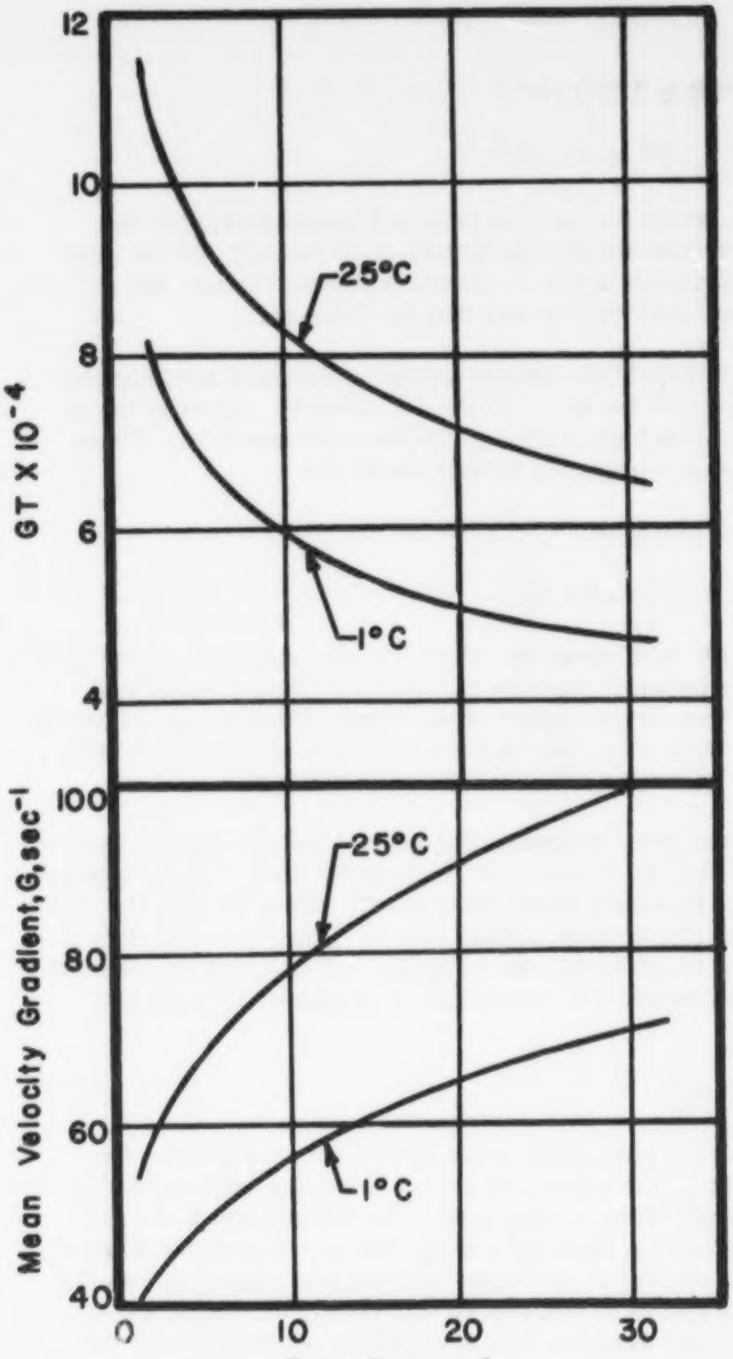


Figure 22. Calculated Values of G and GT as a Function of Flow Rate for the Slow Mixing Portion of the Republic Mine Effluent Channel.

$$G = 700 \text{ to } 1000 \text{ sec}^{-1}$$

$$GT = 2 \times 10^4 \text{ to } 3 \times 10^4$$

Figure 21 indicates that, except for low flow rates and low temperatures, the rapid mix values of G were considerably higher than recommended. On the other hand, because of the short residence time in the steep channel section, the rapid mix values of GT were considerably less than recommended.

Figures 23 and 24 show relationships between jar test supernatant turbidity and slow mixing G and GT values for the Republic Pond 3 effluent. Values of the jar test velocity gradient were determined by using Camp's (11) correlation. These figures indicate that, for slow mixing, the optimal ranges are:

$$G = 35 \text{ to } 48 \text{ sec}^{-1}$$

$$GT = 3 \times 10^4 \text{ to } 6 \times 10^4$$

Comparing these values with those shown on Figure 22 indicates that, except for low flow rates and low temperatures, the slow mix values of G were higher than optimal. Likewise, over most flow and temperature ranges, the slow mix GT values were higher than optimal. However, over the most critical range, (high flow rates and low temperatures) the GT values fell within the optimal range.

Based upon the foregoing analysis, it appears that there is room for improvement in the existing coagulation-flocculation system at the Republic mine. These improvements would consist of modification to bring the G and GT values for both the slow and rapid mixing operations closer to the optimal values. Since the hydraulics of the present system are largely set by the natural terrain, a significant improvement in G and GT would probably require the construction of a mechanical rapid mix-flocculation system.

Handling of Coagulated Solids

As indicated in Section IV, the coagulated solids settled out upon entering the Republic Mine tertiary pond. This caused the accumulation of a large delta of unconsolidated slurry near the mouth of the tertiary pond. The solids content of this slurry varied from about 15% at the top to 30% at the bottom. This material presented a problem as it was situated in the stream path and was subject to scour at high flows. Moreover, the volume of this deposit was rapidly increasing and encroaching upon the pond volume available for clarification. An investigation of thickening and filtration characteristics was conducted to assess the feasibility of collecting and dewatering this and any future solids accumulation. Field surveys showed that, on May 13, 1971, the unconsolidated muck in the tertiary pond

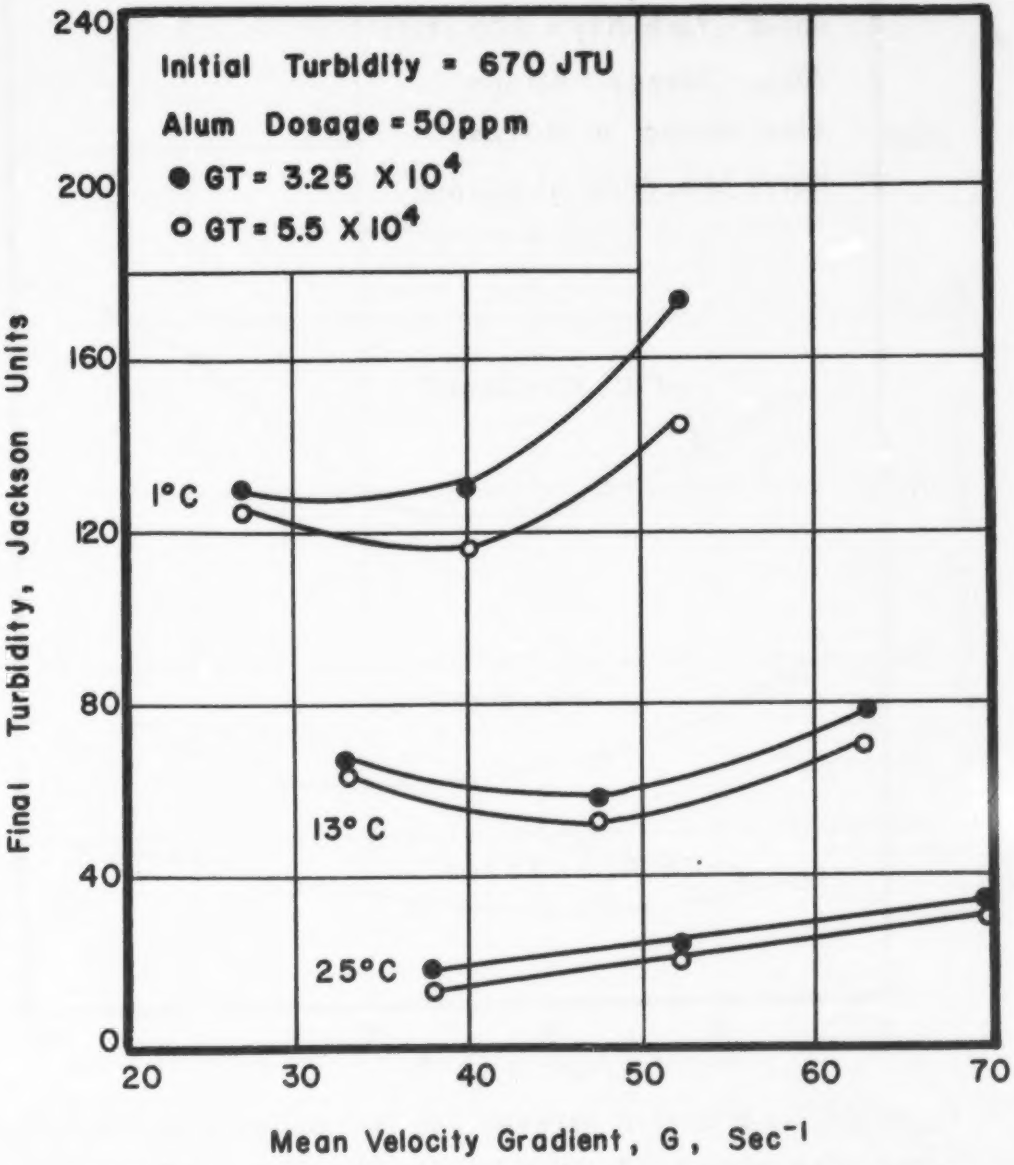


Figure 23. Relationship between Jar Test Supernatant Turbidity and Slow Mixing G Value for Republic Pond 3 Effluent.

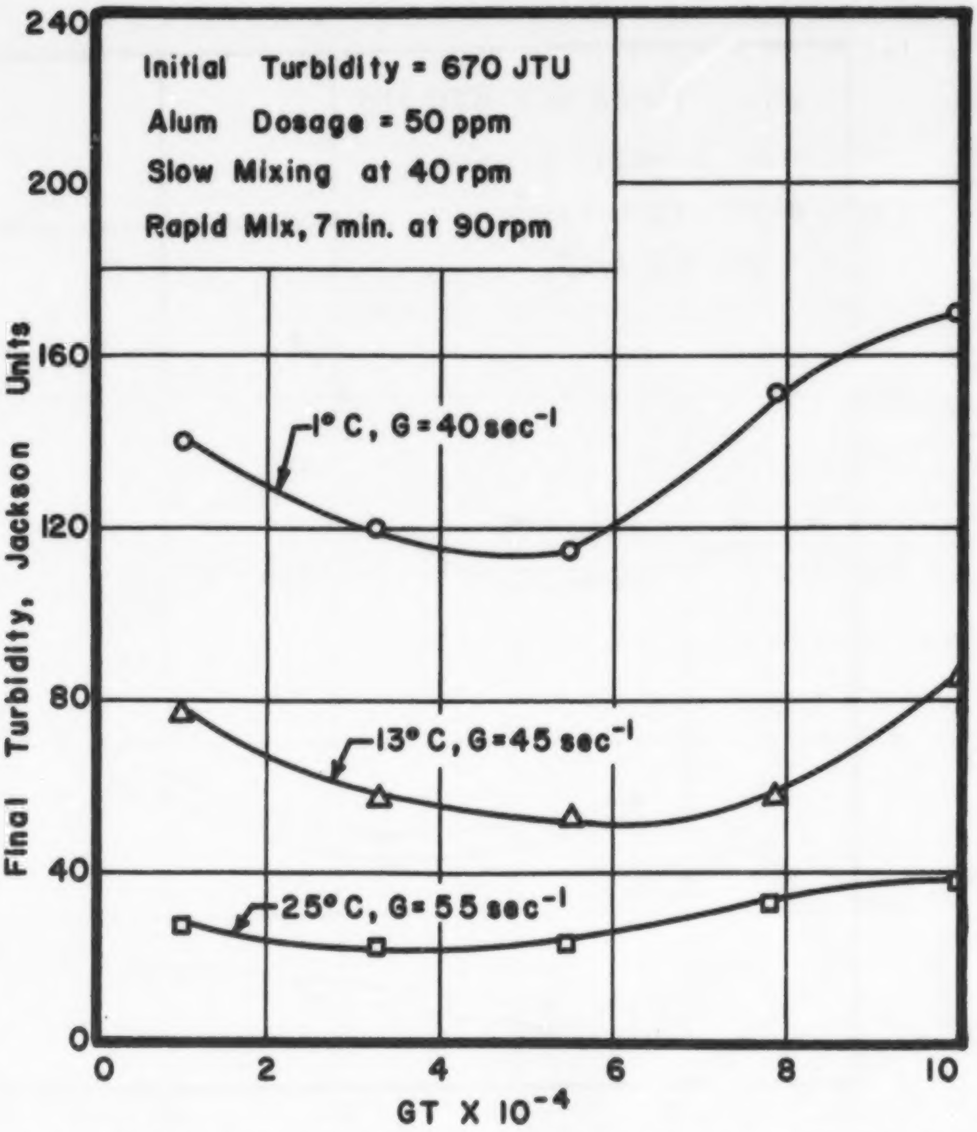


Figure 24. Relationship between Jar Test Supernatant Turbidity and Slow Mixing GT Value for Republic Pond 3 Effluent.

occupied a total of 1,438,100 cubic feet. The average density of the muck was 71.6 lb/ft^3 and the average solids content was 22.6%. The total weight of dry solids contained in the unconsolidated deposit, therefore, amounted to approximately 23.28 x 10^6 lb. or 11,650 tons. Over the period of June 1970 to June 1971, 6,170 tons accumulated.

The unconsolidated slurry employed in both the thickening and filtration experiments was obtained from the deposit existing in the Tertiary Pond. Thus, the material contained some residual alum and polymer. (The quantity of alum added ahead of the tertiary pond amounted to about 5% of the weight of solids treated; the quantity of polymer, Nalco, Na-603, amounted to about 0.5% of the weight of solids treated.) Because of this, the results of both the interface subsidence and filtration experiments may not be applicable to a sludge generated by other coagulants.

Thickening Characteristics

To determine the thickening capacity of the unconsolidated muck, several interface subsidence experiments were performed using 1000 ml graduated cylinders. The control variables in these experiments were:

1. Initial solids concentration, 5%, 10%, 20%.
2. Amount of slow stirring, none, 4 revolutions per hour.
3. Addition of thickening aids.

$FeCl_3$ 3%, 6%, 9% (based on wt. of solids)

Polygalacturonic Acid 0.1%, 0.5%, 1%

CCM-510 polymer 0.1%, 0.5%, 1.0%

CCM-560 polymer 0.01%, 0.05%, 0.1%, 0.5%, 1.0%

In all cases, the direct response measured was the interface subsidence curve. Unit areas were calculated for the conditions which gave the most promising interface subsidence curves. All interface subsidence tests were conducted at room temperature.

Slow stirring or slow raking action was achieved by using an apparatus similar to that described by Eckenfelder (12). The stirrers were powered by synchron motors which revolved at 4 rev/hr.

Figures 25 and 26 show interface subsidence curves for initial concentrations of 10 and 20% with and without slow stirring. The final solids concentrations attained

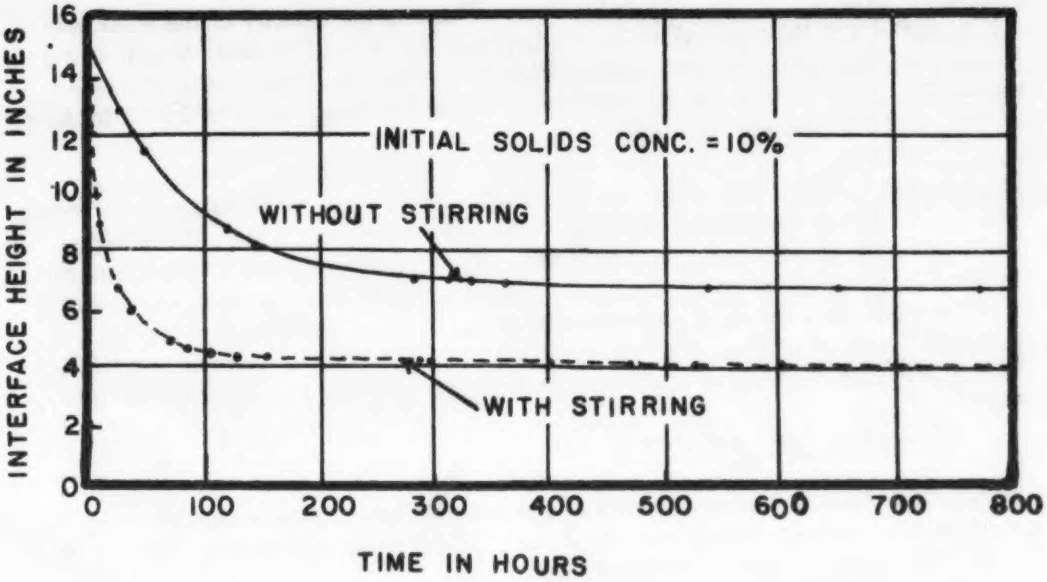


Figure 25. Interface Subsidence Curves for Coagulated Solids Generated at Republic.

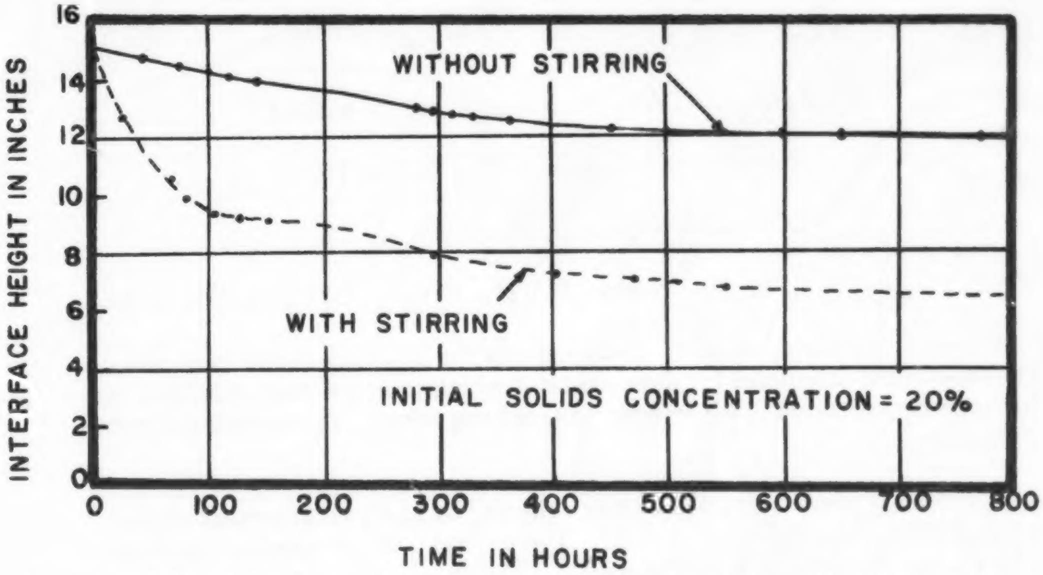


Figure 26. Interface Subsidence Curves for Coagulated Solids Generated at Republic.

at 800 hr. are summarized in Table 11.

Table 11. Solids Concentrations Attained in Batch Thickening Tests

Initial Solids %	Final Solids % No Stirring	Final Solids % Stirring
5%	17%	19.5%
10%	22.5%	35%
20%	24%.	41%

Following the minimum solids flux method outlined by Dick (13) it was possible to estimate thickener unit areas required for various underflow concentrations from the interface subsidence curves. Estimates of the unit areas required to attain underflow concentrations of 20% and 30% are shown in Table 12.

Table 12. Estimate of Unit Areas, ft^2/lb/sec , No Thickening Aid

	Feed Conc. %			
	10%		20%	
Underflow Concentration	Stirred	Not Stirred	Stirred	Not Stirred
20%	9,100	117,000	-	-
30%	90,000	-	32,300	-

Influence of Thickening Aids

A series of interface subsidence experiments was conducted to investigate the effect of various thickening aids. Here, the concentrations of thickening aids employed are expressed as a percentage of the weight of solids. In each experiment, an appropriate amount of a concentrated solution of the thickening aid was added to a 10% suspension of slurry. After gentle mixing the slurry was placed into 1000 ml graduated cylinders and allowed to settle usually under the influence of slow stirring. In several cases, the effect of slow stirring was observed by inserting the stirring apparatus when the interface appeared to reach an equilibrium level in the absence of stirring.

Ferric Chloride: As a thickening aid, ferric chloride was ineffective at concentrations of 3%, 6%, and 9%. It may have been that any additional benefit to be gained by adding a tri-valent metal ion was precluded by the alum residual present in the slurry.

Polygalacturonic Acid: The investigation of this compound was prompted by a suggestion received from the E.P.A. project representative. This compound was ineffective at concentrations of 0.1%, 0.5% and 1.0%.

Cationic Polymer, Calgon M-510: This substance was the most economical coagulant as far as turbidity removal was concerned. In addition, it was the most effective of the vacuum filtration aids tested. However, it was only slightly effective as a thickening aid at concentrations of 1%, 0.5% and 0.1%. Under quiescent conditions, the initial rate of interface subsidence, for concentrations of 0.5% and 1.0%,was increased markedly over that of the control. However, the interface reached an equilibrium height of 10 inches at 50 hours. Further subsidence took place only when stirring was applied at 140 hours. The raking action resulted in an interface subsidence pattern very similar to that of the stirred control. Overall, this substance did not produce a significant improvement over the interface subsidence pattern of the stirred control.

Anionic Polymer, Calgon M-560: This polymer was selected for investigation as a thickening aid on the basis of visual observations made during coagulant jar tests. Overall, the Calgon M-560 polymer was the most effective thickening aid tested. Figure 27 shows the effect of this substance on the interface subsidence curves at concentrations of 0.01%, and 0.5%.

At an initial solids content of 10%, the maximum sludge concentration of approximately 35% was reached. This was comparable to that attained in the corresponding experiment with no thickening aid. At concentrations of 0.1% and above, this polymer caused serious "clumping" in the stirred thickening cylinders. This clumping interfered with the raking action as the clumps tended to move ahead of the raking mechanism. A higher stirring speed tended to eliminate this problem. Fortunately, this clumping action did not occur at the 0.05 % level. Thus, 0.05% of Calgon M-560 was taken to be the optimal thickening aid.

Table 13 lists the unit areas determined from the interface subsidence curve.

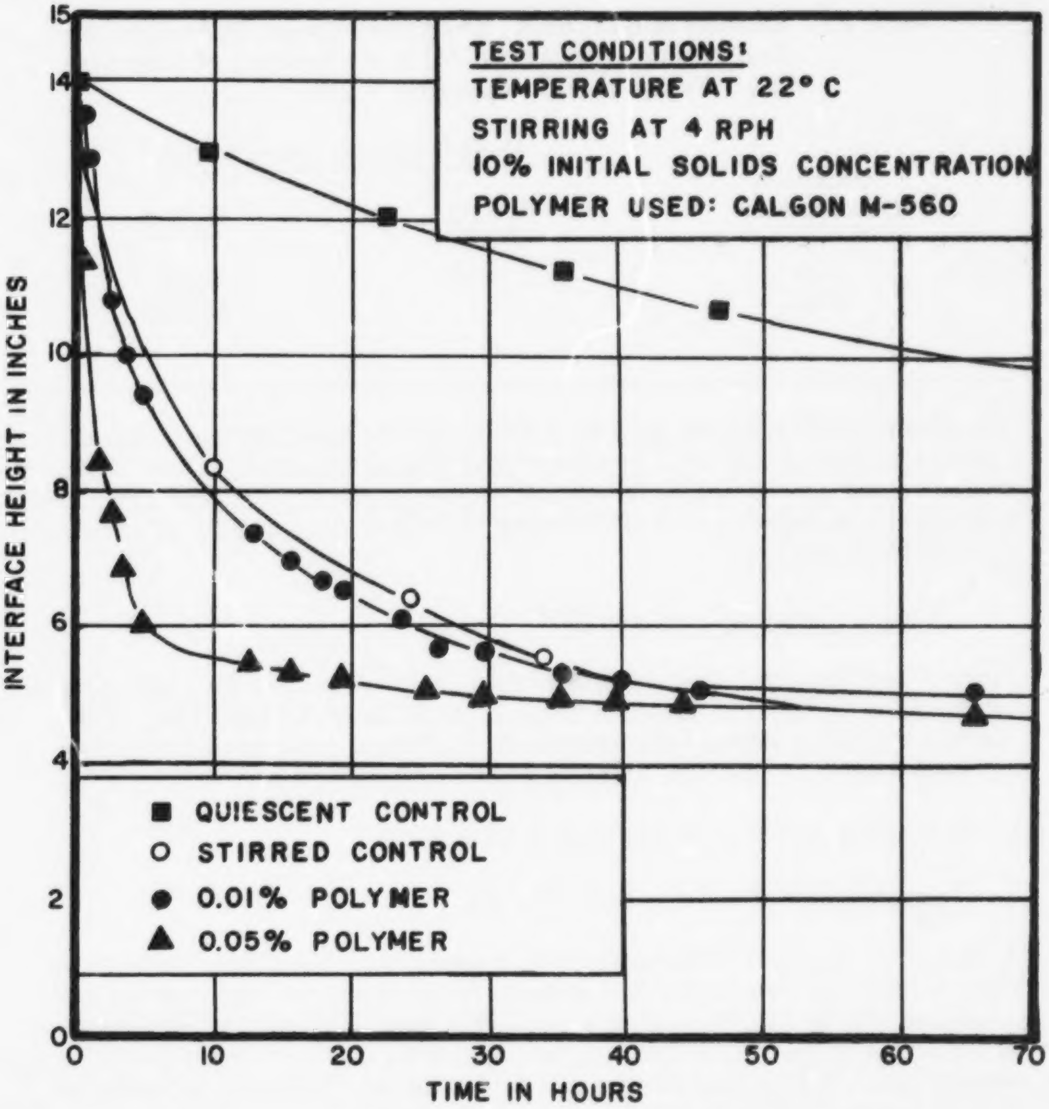


Figure 27. Influence of Anionic Polymer, Calgon M-560, on the Interface Subsidence Pattern of Coagulated Solids Generated at Republic.

Table 13. Estimate at Unit Areas, ft^2/lb/sec, Calgon M-560 Polymer Employed as a Thickening Aid in the Presence of Slow Stirring. Feed Solids Concentration = 10%.

Underflow Concentration	M-560 Polymer Concentration 0.01%	 0.05%
20%	8,400	2,080
30%	----	9,530

The beneficial effect of this polymer is easily seen by comparing the unit areas listed in Tables 12 and 13. To thicken from 10% to 20% under stirred conditions required a unit area of 9,100 ft^2/lb/sec without thickening aid. The same volume reduction in the presence of 0.05% Calgon M-560 required a unit area at only 2,080 ft^2/lb/sec.

Vacuum Filtration Characteristics

Büchner Test Results. The vacuum filtration characteristics of the slurry were investigated using both the Büchner Funnel Test and the Filter Leaf Test. The Büchner test was conducted according to the procedures outlined by Eckenfelder (12) and Rich (3). The chief objectives of these tests were:

1. To evaluate certain constants used in filter design.

2. To evaluate the effect of various filter aids.

A total of 97 individual Büchner tests were conducted. The control variables in these experiments were feed solids content, filtration vacuum, and type and concentration of filter aid. During each test, the volume of filtrate, V, was recorded as a function of time, t. A plot of (t/V) versus V for each test normally yielded a straight line. The slope of the line was then measured graphically and employed to calculate the specific resistance of the filter mat (12).

The observed effects of filter aids on the specific resistance of the filter cake are summarized in Table 14. The feed solids concentration used in these experiments was 10%. In the case of ferric chloride, specific resistance was determined for filter aid concentrations of 3%, 6% and 9%, whereas, for the polymers, specific resistance was normally determined at dosages of 0.01%, 0.05%, 0.10%, 0.25%, 0.5%, and 1%. For the aids applied singly, and for the equal dosages of M-510 and M-560, the values listed in Table 14 represent the optimal dosage.

Table 14. Effects of Filter Aids: Summary of Specific Resistance Values at Various Filter Aid Dosages

Filter Aid	Dosage of Filter Aid (% based on wt. of solids)	Specific Resistance at Given Filter Aid Dosage, sec^2/g
NONE	-	24.8×10^7
Ferric Chloride	3%	10.64×10^7
Cationic Polymer Calgon M510	1%	6.01×10^7
Non-Ionic Polymer Calgon M550	0.5%	8.58×10^7
Anionic Polymer Calgon M560	0.05%	14.9×10^7
Combination		
Calgon M510	0.25%	
Calgon M560	0.25%	4.02×10^7
Combination		
Calgon M510	0.2%	
Calgon M560	0.05%	18.8×10^7
Combination		
Calgon M510	0.4%	
Calgon M560	0.05%	11.75×10^7

In the thickening experiments, the non-ionic polymer Calgon M-560 was found to be the most effective aid at a concentration of 0.05%. When used alone, as a filter aid, the same concentration produced some improvement in specific resistance. Higher concentrations, however, were ineffective. On the other hand, the cationic polymer, Calgon M-510, was ineffective in thickening but was extremely effective in reducing specific resistance. To investigate the effect which might result by adding M-510 to a sludge which had been thickened using M-560, a series of experiments was conducted in which the two polymers were added in equal amounts. The corresponding curve shown in Figure 28 indicates a synergistic effect. This suggested that it would be practical to use the M-560 in a thickening process ahead of a filtration process employing M-510.

Leaf Test Results. To determine the practicality of dewatering the sludge by vacuum filtration, a total of 53 filter leaf tests were conducted using an Eimco Filter Leaf Test Kit. The control variables in these experiments were: the filter fabric, filter aid and concentration, form time and dry time. In nearly all experiments, the feed solids content was maintained at 20%. The general test procedure followed that outlined by Eckenfelder (12).

The results of these experiments are summarized in Table 15. Of the various fabrics supplied with the test kit, the nylon fabric (Eimco No. NY 317F) was judged to be the best on the basis of percent solids remaining in the filtrate and ease of cake removal. It was found that a slight positive air pressure greatly aided in cake removal. On the basis of these experiments, the optimal form time was judged to be 30 seconds. Likewise, the minimum dry time was established to be 30 seconds. Allowing 30 seconds for cake removal would give a cycle time of 1.5 minutes at 33% submergence. Based on a 1.5 minute cycle time and 20% feed concentration, the filter loading rates calculated directly from the leaf test results ranged from 13.9 to 28.1 lb/hr/ft^2 for the M-510 used alone, from 12.3 to 20.1 lb/hr/ft^2 for the various combinations of M-560 and M-510 and from 10.3 to 12.6 lb/hr/ft^2 with no filter aid.

Drying Characteristics in Air

Several drying experiments were conducted in still air under controlled temperature and humidity conditions employing a sludge slurry with an initial solids content of 20%. The drying pans used in these tests contained a layer of coarse sand and were equipped with drains so that gravity dewatering could be distinguished from drying. Slurry depths of 0.5 in. and 1.0 in. were used. The temperature and humidity conditons were: 37°F at 95% R.H. and 64°F at 40% R.H. Figure 29 shows the results of these tests. Gravity dewatering was effective only over the first 25 hours. From 25 hours to 50 hours, the drying rate generally decreased until shrinkage caused large surface cracks to develop. This aided the drying process by increasing the effective contact area.

Field drying experiments were conducted adjacent to a fully instrumented weather station. In these tests, both covered and uncovered pans were employed. The covers allowed circulation of air but eliminated precipitation. The results of these tests showed that precipitation caused the solids in the uncovered pans to liquify and seal the sand surface. Subsequent drying was hindered because large shrinkage

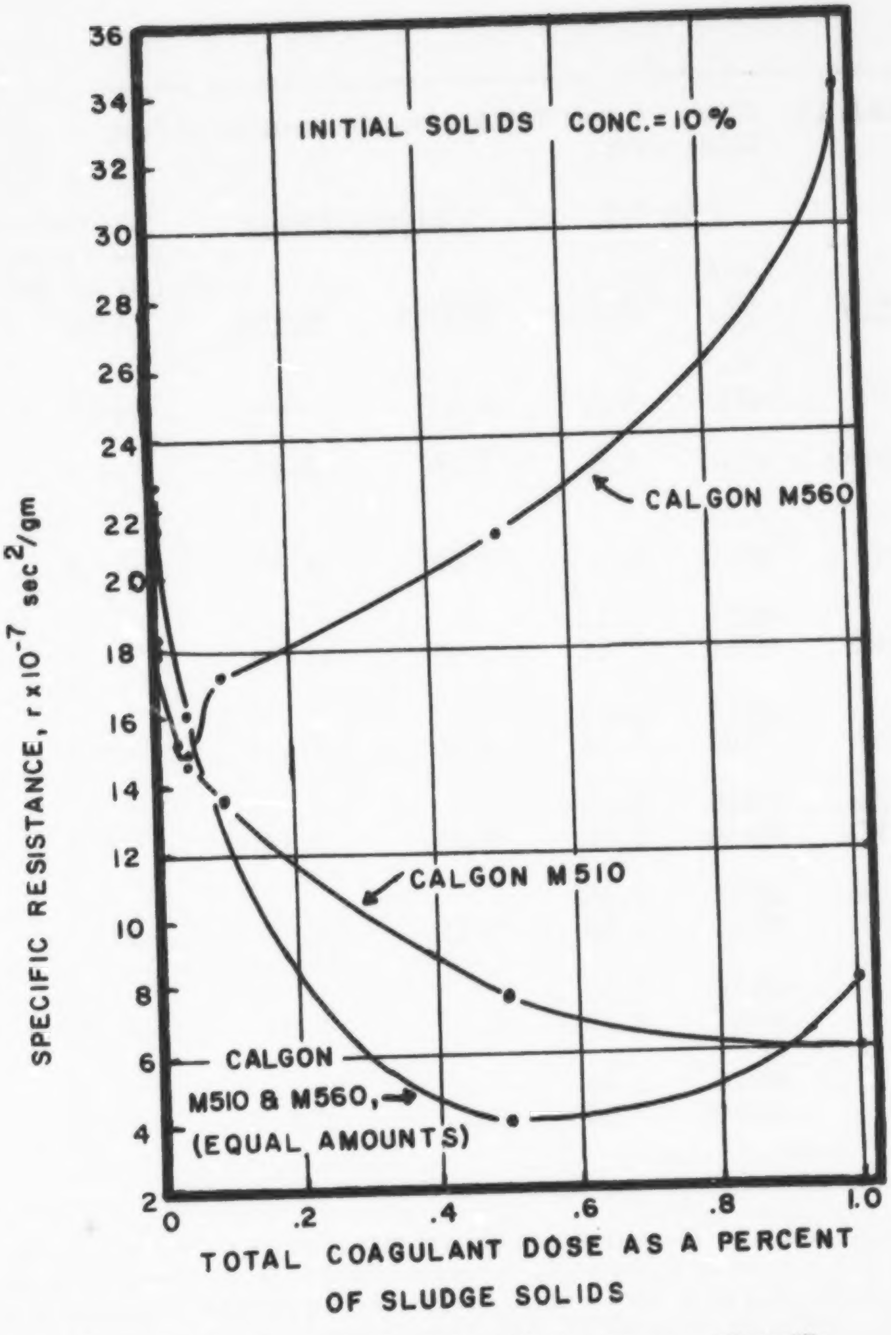


Figure 28. Influence of Filter Aid Dosage on Specific Resistance of Filter Cake.

Table 15. Summary of Leaf Test Results, Concentration of Feed Solids = 20%

	Filter Aid		% Solids in Filtrate		% Solids in cake with aid (30 sec. dry time)
Fabric (Eimco No.)	Calgon No.	Conc.%	with aid	w/o aid	
Nylon (NY 317F)	M510	0.5	0.19	-----	40.7
Polypropylene	M510	0.5	0.25	0.40	42.6
Dynel (CY 453)	M510	0.5	0.35	0.80	39.9
Nylon (NY 317F)	M510 M560	0.15 0.15	0.05	-----	44.8
Nylon (NY 317F)	M510 M560	0.05 0.05	0.10	-----	49.3
Polypropylene (popr 913F)	M510 M560	0.15 0.15	0.05	0.40	42.0
Polypropylene (popr 913F)	M510 M560	0.05 0.05	0.17	0.40	46.9
Dynel (DY453)	M510 M560	0.15 0.15	0.07	0.8	42.5
Dynel (DY453)	M510 M560	0.05 0.05	0.17	0.8	45.0

cracks did not develop. These results suggested that effective air drying would be attainable only on covered drying beds. A mechanical heat-drying operation appears to be more practical if some use for which a relatively pure material is required to be made of the dried solids.

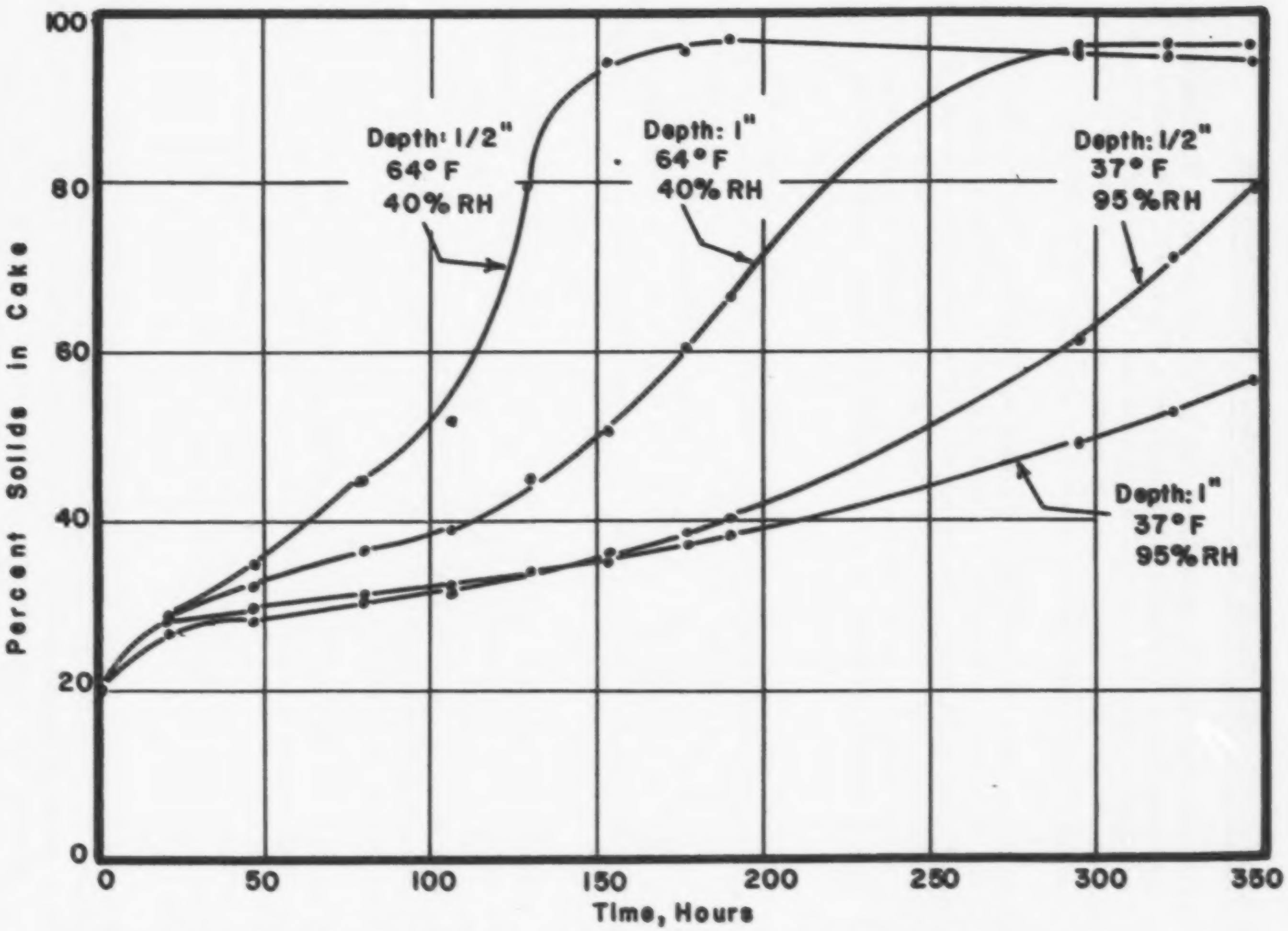


Figure 29. Gravity Dewatering and Drying of Coagulated Solids.

Feasibility of a Possible Solids Handling System at the Republic Mine

The results obtained above suggest that it may be feasible to handle the coagulated solids by a process system similar to that outlined in Figure 30. The data show that it definitely is possible to coagulate, thicken and filter the solids. Although no experimental data were obtained relative to heat-drying, the results of the air drying experiments suggest that mechanical heat-drying would be possible.

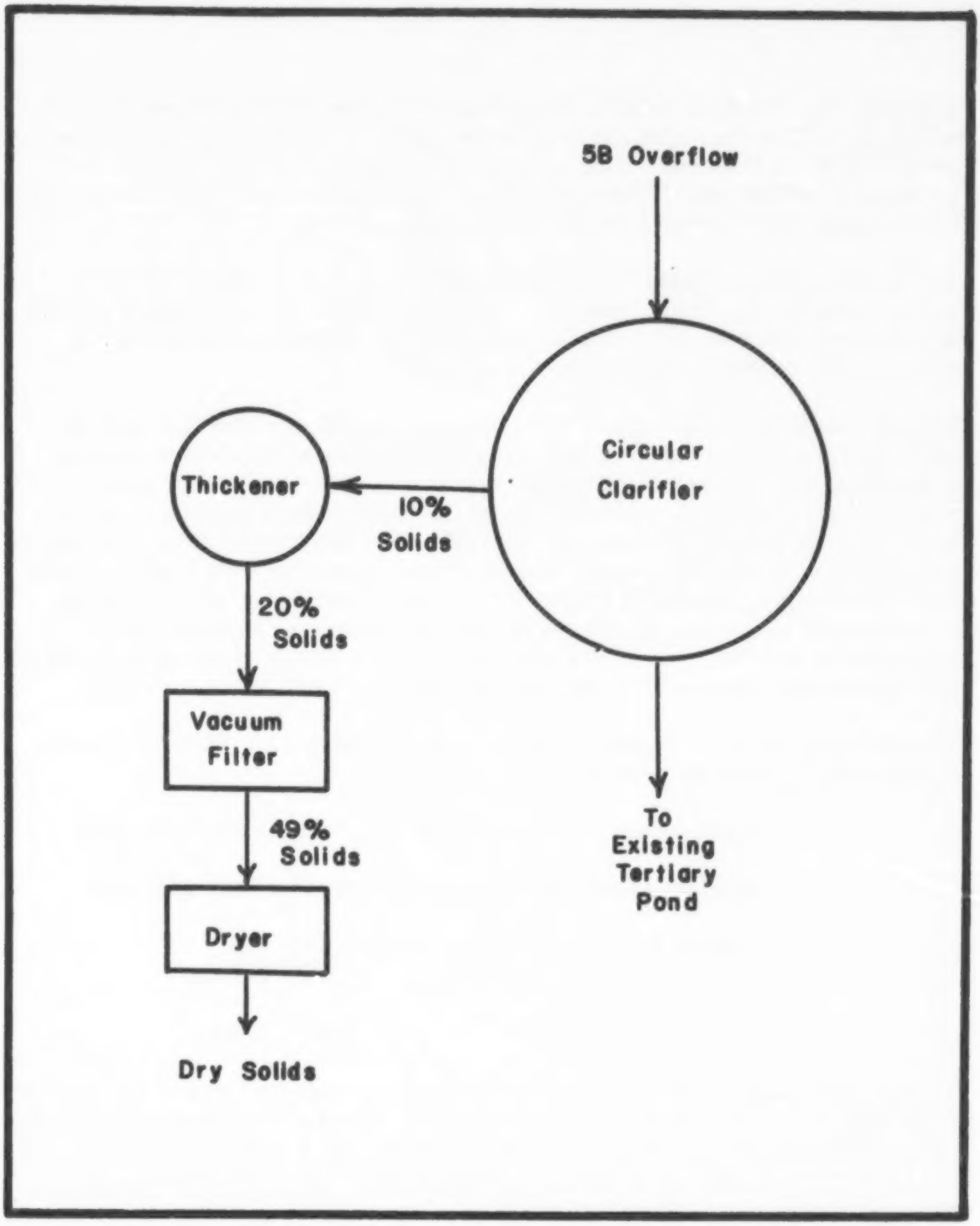
Based on the results of the settling, thickening and filtration characteristics of the solids, reasonable estimates of the areas required for settling, thickening and vacuum filtration can be made. For design of a full scale system, however, more accurate estimates might be made based on continuous flow pilot studies. Nevertheless, these reasonable estimates can be used in conjunction with available cost information to roughly estimate the average annual cost of this solids handling system.

In synthesizing this system, it was assumed that water quality requirements for reuse prohibited both the addition of coagulents within the reuse system and increasing the average percentage of process water recycled. It was also assumed that storage within the impoundment system would be sufficient to produce a nearly constant effluent flow rate close to the average annual rate of 14.7 cfs observed over the study period and that coagulated solids would be generated at the average annual rate of 6170 tons/yr. or 16.9 tons/day observed over the study period.

As previously stated, the coagulated solids employed in the thickening and filtration experiments were obtained from the Tertiary pond and contained some residual alum and Nalco, N_a-603 polymer. Therefore, it was assumed that the same coagulants would be employed ahead of the system shown in Figure 30. Also, since coagulation is presently practiced, the cost of coagulant chemicals was not included in the cost estimate.

The quantities of alum and polymer added amounted to approximately 5% and 0.5%, respectively, of the weight of solids treated. At the average initial solids content (427 ppm, 850 JTU) this dosage resulted in a chemical cost of $9.60/MG or $5.41/ton of solids. If coagulated solids produced by Calgon M-510 thickened adequately, a 40% to 60% saving in chemical cost might be realized (see Figures 17 and 20).

The cost estimates developed in the following section have been adjusted to reflect September 1971 levels. In most instances, this adjustment was based on the Engineering News Record Building Cost Index. (For Sept. 1971 the ENR Building Cost Index = 990) Capital Costs have been converted to average annual costs using a capital recovery factor of 0.11683 based upon an interest rate of 8% and an amortization period of 15 years. Ten percent of the first cost was assumed to cover engineering costs.



5B Overflow
Circular Clarifier
Thickener
10% Solids
20% Solids
Vacuum Filter
49% Solids
Dryer
Dry Solids
To Existing Tertiary Pond

Figure 30. Possible System for Handling Coagulated Solids at the Republic Mine.

Clarification: The chief purpose of the circular clarifier shown in Figure 30 is to remove the bulk of the solids and to allow the deposited sludge to be collected and transferred to the thickener. This operation would be located near the inlet to the existing tertiary pond. Since the effluent would subsequently pass through the tertiary pond, a conservative design would not be required.

In a properly designed basin it might be possible to collect the deposited solids without a mechanical sludge collector. This could result in a considerable savings in first cost. However, the following analysis is based upon a circular clarifier equipped with mechanical sludge collection.

Settling column analyses of the 5-B overflow coagulated with alum (25 ppm) and Nalco Na 603 (2 ppm) indicated that the effluent concentration could be reduced to about 50 ppm with detention time of about 1.67 hr. in a basin 6 ft. deep. (Overflow rate = 650 gal/day/ft^2). The sludge concentration attained in the bottom of the column after 1.67 hr. was about 5%. For this type of flocculent settling a deeper basin would give a higher removal efficiency and facilitate sludge thickening. Hence it was assumed that a basin 10 ft. deep operated at an overflow rate of 650 gdp/ft^2 would give the desired removal and produce an underflow solids concentration of 10%. The clarification area required is then estimated as 1.46×10^4 square feet, equivalent to two circular clarifiers, each 96 ft. in diameter.

The unit cost, based on September 1971, is estimated at \$25.70/ft^2 including appurtenances and engineering (14). Thus:

$$\text{Capital Cost} = 1.46 \times 10^4 ft^2 (\$25.70/ft^2) = \$373,000$$

$$\text{Annual Capital Cost} = \$373,000 (0.11683) = \$43,800$$

$$\text{Capital Cost per Ton of Dry Solids} = \frac{\$43,800}{6,170 \text{ Tons}}$$

$$'' \qquad = \$7.10/\text{ton}$$

Thickening: Since the solids content of the thickener underflow affects the required filter area, the thickening and filtration operations must be considered together. A comparative cost analysis of both operations showed that it is more economical to thicken to 20% solids than to thicken to 30% solids. Thus, the estimates are based on thickening to 20% solids.

Table 13 shows the required unit area to be 2080 ft^2/lb/sec with 0.05% Caigon M-560. Using an excess capacity or scale-up factor of 2, gives a unit area of 4160 ft^2/lb/sec. The mass rate of solids flow is 6170 tons/yr. or 0.392 lb/sec on a dry basis. Thus,

$$A = 4160 \frac{ft^2}{lb/sec} (0.392 \text{ lb/sec}) = 1635 \text{ ft}^2$$

(1 circular thickener, 46 ft in diameter)

The unit cost, (September 1971) including appurtenances and engineering was estimated (14) as \$52.80 /ft^2 Thus:

$$\text{Capital Cost} = 1635 \text{ ft}^2 (\$52.80/\text{ft}^2) = \$86{,}300$$

$$\text{Annual Capital Cost} = \$86{,}300 \ (0.11683) = \$10{,}060$$

$$\text{Capital Cost/Ton Solids} = \frac{\$10{,}060}{6170 \text{ Ton}} = \$1.63/\text{ton}$$

The Calgon M-560 polymer costs \$1.38 per lb. Thus, the annual chemical cost for thickening would be:

$$6170 \ (2000) \text{ lb/yr} \left(5 \times 10^{-4} \frac{lb}{lb}\right) (\$1.38/\text{lb}) = \$8{,}550$$

or \$1.38 per ton of dry solids.

Filtration: Equation 10 was employed to estimate filter loading rates (12).

$$L = 35.7 \left[\frac{P^{(1-s)} \gamma}{\mu \ R_o} \right]^{0.5} \frac{C^m}{t^n} \qquad 10.$$

where:

L = loading rate, $\frac{lb/ft^2}{hr}$
P = vacuum, psi
μ = filtrate viscosity, centipoise
R_o = specific resistance, sec^2/g x 10^{-7} (determined from Büchner Test) divided by p (vacuum during Büchner Test) raised to the s power.
s = coefficient of cake compressibility (determined from Buchner data by plotting log specific resistance vs log P)
C = cake solids deposited per unit volume of filtrate, g/ml
m = exponent (determine from leaf test data by constructing a log-log plot of loading vs. initial feed solids at constant cycle time)
t = cycle time, minutes
n = exponent (determined from leaf test data by constructing plot of loading vs. form time at constant initial feed solids)

γ = fraction of cycle time devoted to cake formation

The constants m, n and s were evaluated from the laboratory data as:

$m = 0.80$
$n = 0.88$
$s = 0.43$

The filter loading rate was estimated for the following conditions:

Filter Aid: 0.05% Calgon M-510 Polymer
0.05% Calgon M-560 residual from thickening
Feed Solids Content = 20%
Temperature = 55 deg. F.
Vacuum = 9.8 psi
Cycle Time = 1.5 minutes, 0.5 minutes form time

Figure 28 indicates that at dosages of 0.05% Calgon M-560 and 0.05% Calgon M-510, the specific resistance would be no greater than 16×10^7 sec^2/g. Thus, the loading rate can be estimated from Equation 10 as 10.45 $\frac{lb/hr.}{ft^2}$ This value is somewhat more conservative than the values at 12.3 to 20.1 $\frac{lb/ft^2}{hr}$ determined directly from the leaf test data for various combinations of Calgon M-510 and M-560 polymers. In view of this, an excess capacity factor was not applied here.

If it is assumed that the filter operates 6 hr/shift or 18 hr/day, the solids loading rate during operation is 1880 lb/hr and the required filter area is 181 ft^2.

The capital cost of the filter including building, appurtenances and engineering, was estimated (14) as $159,500. The annual capital cost, then is, $159,500 (0.11683) = $18,380. The Calgon M-510 polymer costs $0.44 per lb resulting in a chemical cost of $2,720 per year. The power is estimated as (15) $1,790 per year so that the total capital plus power and chemical cost amounts to $22,890 per year or $3.71 per ton of solids.

Mechanical Heat Drying: A rough estimate of the probable cost of heat drying the filter cake at 49% solids can be made by using literature sources.

It will be assumed here, for purposes of estimating, that a direct heat rotary dryer will suffice. The cost of a direct heat rotary dryer, including buildings, appurtenances and engineering cost was estimated using information from Perry (16) as $252,000. This amounts to $29,040 per year or $4.70 per ton.

The dryer would handle 1410 lb solids/hr and 1475 lb water/hour. Assuming the specific heat of the solids to be 0.2, the initial temperature to be 55°F, and neglecting heat recovery, the heat requirement would be roughly 1.73×10^6 BTU/hr. If the overall thermal efficiency is assumed to be 60% (17), 2.89×10^6 BTU/hr would have to be supplied. If fuel oil with a heat value of 1.4×10^5 BTU/gal can be obtained for $0.10/gallon, the annual fuel cost would be

$$2.89 \times 10^6 \frac{\text{BTU}}{\text{hr}} \; \frac{\$0.10/\text{gal}}{1.4 \times 10^5 \ \text{BTU}/\text{gal}} (365)(24) = \$18,100$$

or $2.93 per ton of dry solids.

For comparison, the fuel cost for drying phosphate slimes at an initial solids content of 30% in a fluid bed operation has been estimated as $2.50 per ton of dry solids (17).

Summary: Solids Handling Cost Estimate: The solids handling cost estimate is summarized in Table 16. The operation and maintenance costs were estimated on the basis of 8,760 hr of operation per year requiring an average of 1.5 men at an average wage cost of $7 per hour per man. An additional $7,500 per year was allowed for maintenance. The total cost of $37.65 per ton compares well with costs encountered in processing sewage sludges.

Ultimate Disposal of Dewatered Solids

In view of the relatively plentiful amount of land available it is questionable whether this system can be justified unless the 16.9 tons/day of dried solids can be shown to possess some economic value. The dried solids, if pulverized, would consist of a red powder. It is conceivable that this material might be useful as a component of concrete building materials, as a soil sealant, as a binding material, or perhaps, as a pigment. Any economic value attached to the dried solids would help to offset the cost of the processing system considered here.

In the absence of any economic value, the dry or semi-dry solids would probably have to be ultimately disposed of by burial. Direct spreading on the land surface would probably not be suitable unless special precautions were taken to prevent resuspension and erosion. If land disposal or burial were to be practiced, drying of the solids would not be necessary and the cost per ton of dry solids would decrease to about $26 per ton (assuming that 25% of the O & M cost is attributable to drying) exclusive of final disposal.

Alternative Strategies for Handling Tailings Basin Overflows

In synthesizing the system discussed above, it was assumed that water quality requirements for reuse prohibited both the addition of coagulants within the reuse

Table 16. Summary of Solids Handling Cost Estimate

	Initial Capital Cost	Average Annual	$/Ton Dry Solids
Clarification and Primary Solids Collection			
Capital Cost	$373,000	$ 43,800	$ 7.10
Thickening			
Capital Cost	86,300	10,060	1.63
Chemical Cost		8,550	1.38
Filtration			
Capital Cost	159,500	18,380	2.98
Power		1,790	0.29
Chemical		2,720	0.44
Heat Drying			
Capital Cost	252,000	29,040	4.70
Fuel		18,100	2.93
Operation and Maintenance		100,000	16.20
TOTAL	$870,800	$232,440	$37.65

system and increasing the average percentage of process water recycled. However, it was pointed out in Section IV that little information is presently available on water quality criteria for reuse. Thus in a particular case, the optimal strategy would depend on the reuse water quality requirements.

The polymeric thickening aid employed with the reuse system at the Empire Mine apparently does not interfere with the magnetic concentration process. If a coagulant were to be added to the Republic tailings upon entering Pond 1, the fine particles could be induced to settle with the coarse fraction and the red water problem might be eliminated. However, before the practicality of this and other schemes involving changes in the recycle system can be assessed, more information on recycle water quality requirements must be developed.

SECTION VI

SETTLING CHARACTERISTICS OF TAILINGS PARTICLES IN NATURAL WATER SYSTEMS

The objective of this aspect of the study was to investigate the physical and transport characteristics of the supra-colloidal particles contained in tailings basin discharges with a view toward predicting the rate at which these particles might be removed from a natural water system.

The research plan employed to achieve this objective relied on experimentation with both the Republic and Empire. Mine tailings. The general approach involved column studies conducted to investigate the effects of diluent water, temperature and alum on settling properties. These studies were complemented by a field study conducted on the Republic Tertiary Pond effluent stream. The purpose of the field study was to compare removals predicted from column data with that occurring in the stream. Finally particle size distributions determined by microscopic techniques were compared with these calculated from gravity settling data and Stokes Law.

Quiescent Column Tests

In these analyses, the tailings water containing the fine particles was diluted by factors of 5 and 10 with various natural waters. Upon dilution, the tailings-natural water mixture was allowed to settle under quiescent conditions. The procedure followed was similar to that described in Appendix B, except that the settling columns were only 30 inches high and had only 4 sampling ports spaced at 6 inch intervals.

The Republic tailings water used in these experiments was obtained from the 5B outlet before alum addition. Since the Empire tailings pond effluent generally contained less than 20 mg/l of suspended solids, the Empire tailings used in these experiments were obtained by presettling a sample of the influent to the Empire Basin until the supernate contained a suspended solids concentration comparable to that attained at Republic. Table 17 summaries the conditions employed during each series of quiescent settling experiments.

Various natural waters were used in the settling analyses to determine the effects, if any, of the natural water source on fine tailings particle setteability. Natural waters were obtained from the following sources:

1. Gambles Creek located near the Republic tailings basins.
2. Tertiary effluent stream located at the Republic tailings basin (contained some residual alum and polymer).

Table 17. Quiescent Settling Column Tests Performed

Series	Fine Tailings Sample	Column No.	Natural Water	Dilution Factor	Water Temp.°F	Alum Added
1	Republic	1	Lake Michigan	1/5	40	none
	"	2	"	1/10	40	"
	"	3	"	1/5	70	"
	"	4	"	1/10	70	"
2	Republic	1	Lake Superior	1/5	40	none
	"	2	"	1/10	40	"
	"	3	"	1/5	70	"
	"	4	"	1/10	70	"
3	Republic	1	Gambles Creek	1/5	40	none
	"	2	"	1/10	40	"
	"	3	Tertiary Effluent	1/5	40	"
	"	4	"	1/10	40	"
4	Empire		(Same as Series 1)			
5	Empire		(Same as Series 2)			
6	Empire	1	Schweitzer Creek	1/5	40	none
	"	2	"	1/10	40	"
7	Republic	1	Lake Michigan	1/5	40	2 ppm
	"	2	"	1/10	40	"
	Empire	3	"	1/5	40	"
	"	4	"	1/10	40	"
8	Republic	1	Lake Superior	1/5	40	"
	"	2	"	1/10	40	"
	Empire	3	"	1/5	40	"
	"	4	"	1/10	40	"

3. Schweitzer Creek located near the Empire tailings basin.
4. Lake Michigan (from the water treatment plant intake at Gladstone, Michigan).
5. Lake Superior (from the water treatment plant intake at Marquette, Michigan).

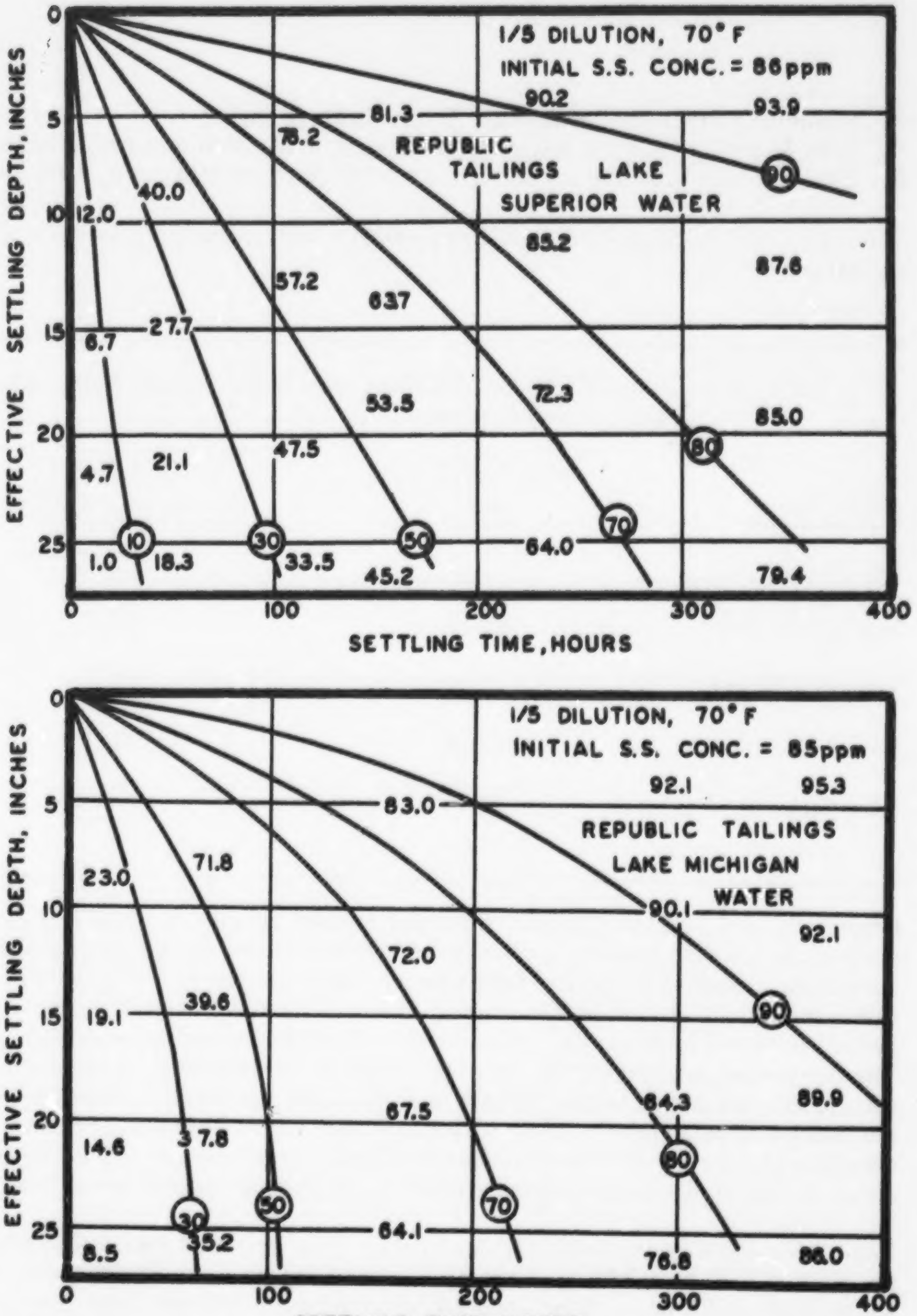
Table 18 gives the dissolved solids concentration and conductivity for each of the natural waters.

Table 18. Dissolved Solids and Conductivity of the Natural Waters Used in the Settling Analyses

Natural Water	Dissolved Solids ppm	Conductivity micromhos/cm@20°C
Gambles Creek	71.2	41.0
Schweitzer Creek	88.0	66.3
Tertiary Effluent	123.0	95.0
Lake Michigan	146.6	122.0
Lake Superior	75.8	56.5

Figures 31 and 32 show typical iso-percent removal curves. Each of these figures summarizes the data obtained from a single settling column experiment. These curves are analogous to the iso-concentration curves presented in Section IV. However, instead of plotting concentrations remaining at a particular time and depth, percent removals based on the initial suspended solids concentration were plotted. The iso-percent removal curves were then drawn by interpolation. Each of the iso-percent removal curves traces the path of a particular size fraction of particles as the fraction is being removed from suspension over a period of time. Downward curvature of the removal lines indicates the flocculating nature of the suspension. A suspension of non-flocculating particles would produce linear removal lines.

Table 19 summaries the settling column data. Total removals were calculated from the plots at 100 hr., 200 hr., and 300 hr. settling times for Republic tailings suspensions and 8 hr., 16 hr., and 24 hrs. for Empire tailings suspension. These total removals represent the percent removals which would occur in an ideal plug flow stream, 27 inches deep after the stated passage times. It has been assumed that particle scour can be neglected. These removals were computed from the corresponding iso-percent removal curves according to the procedure given by Rich (3).



Figures 31(top) and 32(bottom). Typical Iso-Percent Removal Curves for Republic Fine Tailings.

Table 19. Summary of Quiescent Settling Column Data

				Init. S.S.	Total Removal, %		
Natural Water	Tailings	Dilution	Temp°F	conc., ppm	300 hr	200 hr	100 hr
L. Michigan	Republic	1/5	40°	88	80.6	64.1	50.2
L. Michigan	Republic	1/10	40°	45	84.6	76.1	61.1
L. Michigan	Republic	1/5	70°	85	87.6	79.6	66.0
L. Michigan	Republic	1/10	70°	40	91.3	85.3	70.0
L. Superior	Republic	1/5	40°	90	74.2	63.8	45.1
L. Superior	Republic	1/10	40°	44	81.6	74.3	52.0
L. Superior	Republic	1/5	70°	86	85.0	74.8	56.6
L. Superior	Republic	1/10	70°	45	89.8	85.6	65.0
Gambles Cr.	Republic	1/5	40°	90	69.3	52.4	34.9
Gambles Cr.	Republic	1/10	40°	42	77.6	55.9	33.6
Tertiary Effl.	Republic	1/5	40°	80	79.0	65.7	37.6
Tertiary Effl.	Republic	1/10	40°	40	90.4	83.0	56.3
					24 hr	16 hr	8 hr
L. Michigan	Empire	1/5	40°	85	89.3	85.0	76.6
L. Michigan	Empire	1/10	40°	43	87.7	84.7	78.3
L. Michigan	Empire	1/5	70°	85	93.2	92.1	77.5
L. Michigan	Empire	1/10	70°	45	87.8	85.3	74.1
L. Superior	Empire	1/5	40°	90	88.9	85.8	80.2
L. Superior	Empire	1/10	40°	44	87.9	84.0	75.8
L. Superior	Empire	1/5	70°	85	89.4	86.1	80.9
L. Superior	Empire	1/10	70°	43	93.4	88.0	82.1
Schweitzer	Empire	1/5	40°	90	90.2	86.0	76.7
Schweitzer	Empire	1/10	40°	46	88.3	84.7	76.7
	2 ppm Alum Addition				300 hr	200 hr	100 hr
L. Michigan	Republic	1/5	40°	80	77.6	71.6	58.0
L. Michigan	Republic	1/10	40°	40	90.7	81.7	64.5
L. Superior	Republic	1/5	40°	85	82.4	69.1	43.9
L. Superior	Republic	1/10	40°	42	88.1	76.4	57.1
					24 hr	16 hr	8 hr
L. Michigan	Empire	1/5	40°	90	87.7	84.4	70.2
L. Michigan	Empire	1/10	40°	48	88.4	84.1	76.4
L. Superior	Empire	1/5	40°	90	88.5	83.6	78.4
L. Superior	Empire	1/10	40°	43	89.0	83.2	69.4

Analysis of Factors Affecting Tailings Removal

To examine the effects of temperature, dilution, and natural water constituents on the removal efficiency of Republic and Empire fine tailings particles, plots of the total removal versus settling time were made using the data given in Table 18.

Republic Tailings Removal

Figure 33 shows the effect of temperature and dilution on the removal of Republic fine tailings in Lake Michigan and Lake Superior water. The overall effect was an increase in total removal with an increase in the water temperature. However, the increase in total suspended solids removal cannot be entirely accounted for by the decrease in fluid viscosity alone. Increased settling rates apparently resulted from a greater tendency for particles to flocculate at high temperatures. For example, for the settling of Republic fine tailings at a 1/5 dilution in Lake Michigan water, it took approximately 250 hours for 50% of the particles to settle through a depth of 27 inches at 40 degrees. The same removal through the same depth required only 100 hours at 70°F. This represents a 150% increase in settling velocity. Stokes Law predicts that the decreased viscosity would cause the settling velocity to increase by only 52.5%. Therefore, the increase in settling velocity caused by the decrease in viscosity only accounts for approximately 35% of the total increase. The effect of temperature at a dilution factor of 1/10 was similar to the 1/5 dilution runs. However, the differences between the temperature curves were slightly less pronounced.

Figure 34 shows the effect of dilution on the removal of Republic fine tailings from several natural waters. In nearly all cases, increased percent removals occurred at increased dilutions. The effect of dilution was the greatest for the tertiary effluent stream and the smallest for Gambles Creek water. Interpreting this effect in light of Table 18 suggests that it was related to the dissolved solids content of the diluent water. Figure 35 compares the effects of various diluent waters and offers further evidence that the dissolved solids content of the diluent water is directly related to the percent removal obtained. It is reasonable to expect this since the dissolved solids would act to lower the zeta potential of the negatively charged particles. The Tertiary Effluent Stream (D.S.=123 ppm) tended to produce higher removals than Lake Michigan (D.S.=146.6 ppm). This was probably a result of the residual concentration of alum and polymer remaining in the Tertiary Effluent Stream.

Empire Tailings Removal

The Empire fine tailings were removed from suspension at a much faster rate than Republic fine tailings particles. For example, after 32 hours settling time, over

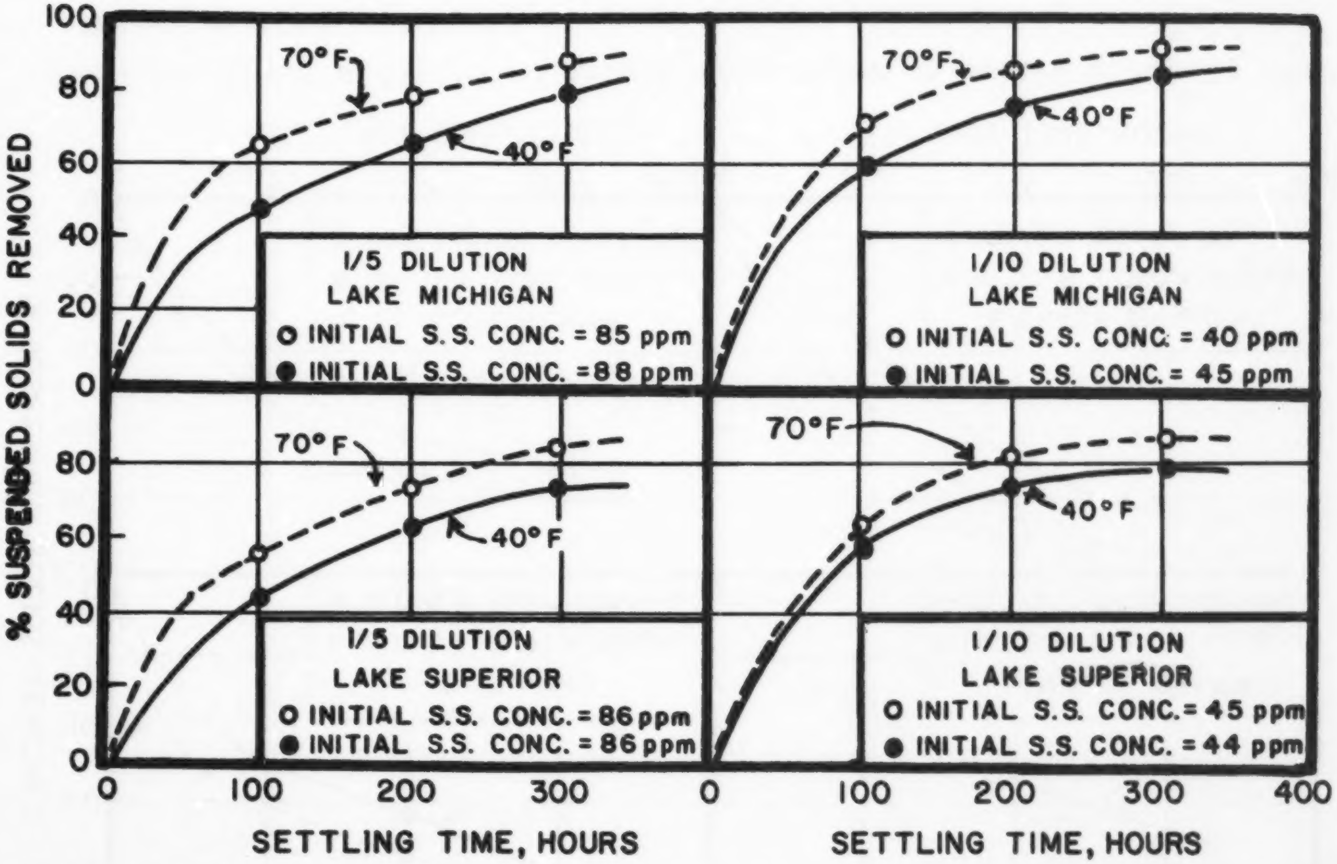


Figure 33. Effect of Temperature and Dilution on Removal of Republic Fine Tailings in Lake Michigan and Lake Superior Water.

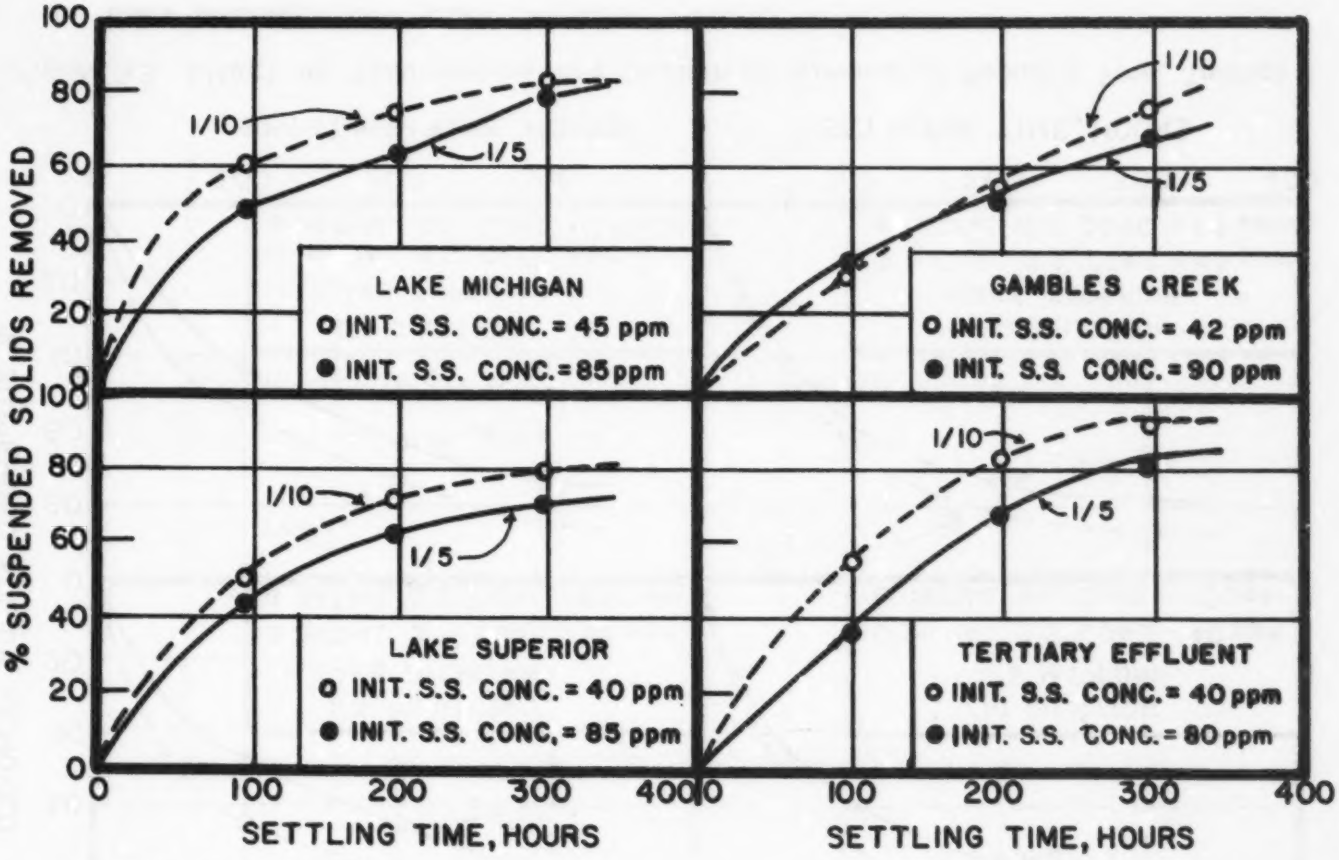


Figure 34. Effect of Dilution and Natural Water Source on Removal of Republic Fine Tailings at 40° F.

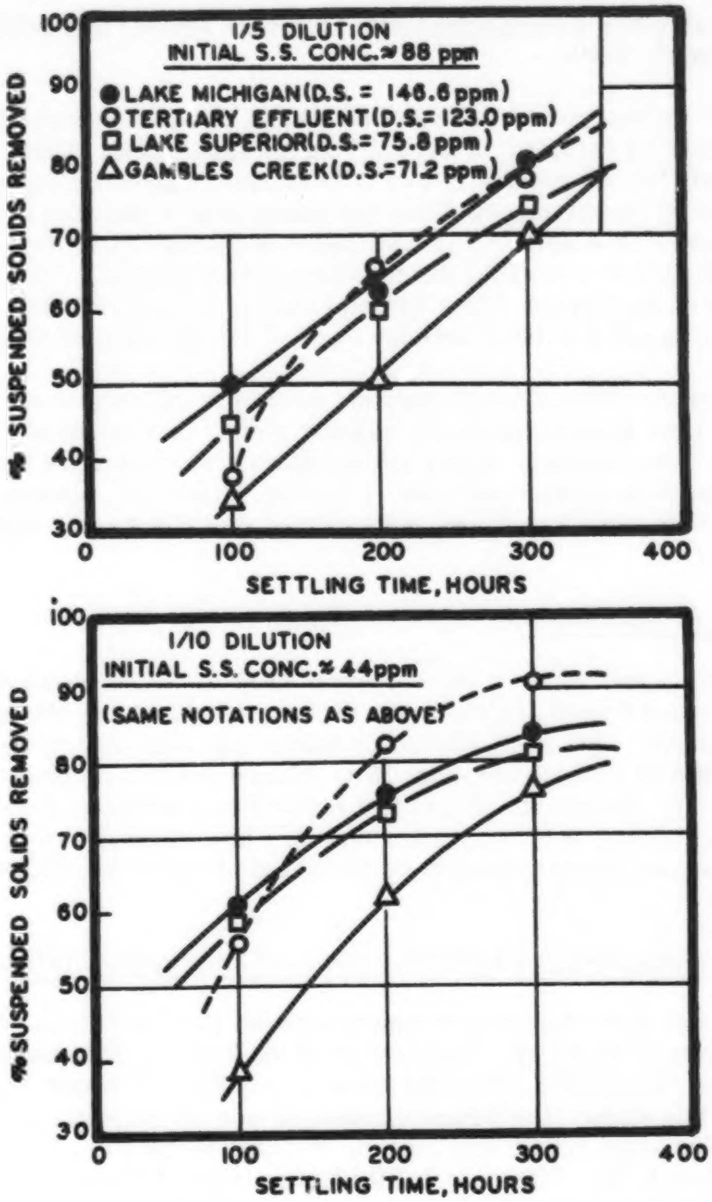


Figure 35. Effect of Various Diluent Waters on the Removal of Republic Fine Tailings at 40°F.

80% removal took place in the Empire tests. The same removal took 300 to 400 hours in the Republic tests.

The iso-percent removal plots for Empire fine tailings particles indicate predominately discrete settling especially at the 40°F temperature levels. Thus, it appears that the relatively fast removals of Empire tailings over Republic tailings would not be accounted for by increased coagulation and flocculation of particles of comparable size. However, the particle size distribution of Empire fine tailings indicated a wider range of particle sizes than that for Republic fine tailings. Also, the specific gravity of the Empire tailings particles used in the analyses was greater than that for the Republic tailings particles (i.e., 2.63 as compared to 2.00).

Figure 36 shows the effect of temperature and dilution on the removal of Empire fine tailings in Lake Michigan and Lake Superior water. Comparison of this figure with Figure 33 (note difference in time scales) shows that temperature exerted less effect on removal of the Empire particles. Likewise, Figure 37 indicates that the dissolved solids content of the diluent water exerted less effect on removal of the Empire particles.

Influence of Alum Addition

Several additional experiments in which 2 ppm of alum was added were conducted on both Republic and Empire fine tailings particles using both Lake Michigan and Lake Superior water. This small alum concentration was used to simulate what might happen when an effluent containing 10 to 20 ppm of alum was diluted by factors of 5 or 10. The results showed that alum slightly increased the removals for Republic fine tailings in both Lake Michigan and Lake Superior waters by about 0 to 10%. However, no consistent effect could be detected for the Empire fine particles.

Field Study: Tailings Removal in the Republic Mine Tertiary Pond Effluent Stream

The purpose of this field study was to compare removal predicted from column data with that occurring in the stream. The section of the Republic Mine tertiary effluent stream below the tertiary pond outlet and above County Road FFW (see Figure 3) was chosen for this study. The stream distance between these points is about 3,800 feet. The upper 2,000 feet consists of a channel roughly 10 feet wide with a velocity of about 1 fps. The lower 1,800 feet consists of an area of impounded water with a much lower velocity. Station 1 was located at County Road FFW, Station 3 at the Tertiary Pond outlet and Station 2 was located 2,000 feet below Station 3.

A dye dispersion test was performed to determine the mean residence time between Stations 3, 2 and 1. At the same time replicate suspended solids analyses were performed at these stations and samples were taken at Station 3 and subjected to

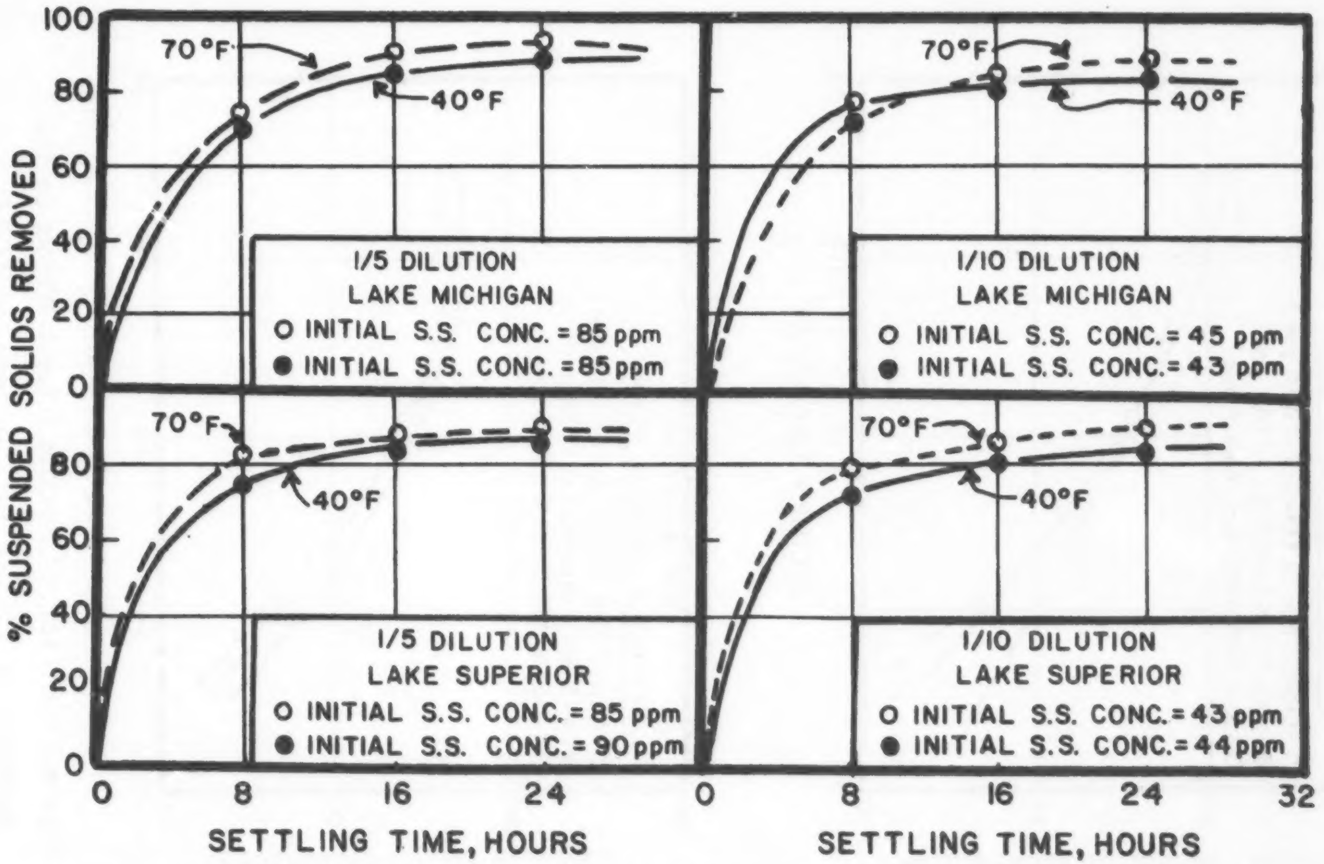


Figure 36. Effect of Temperature and Dilution on Removal of Empire Fine Tailings in Lake Michigan and Lake Superior Water.

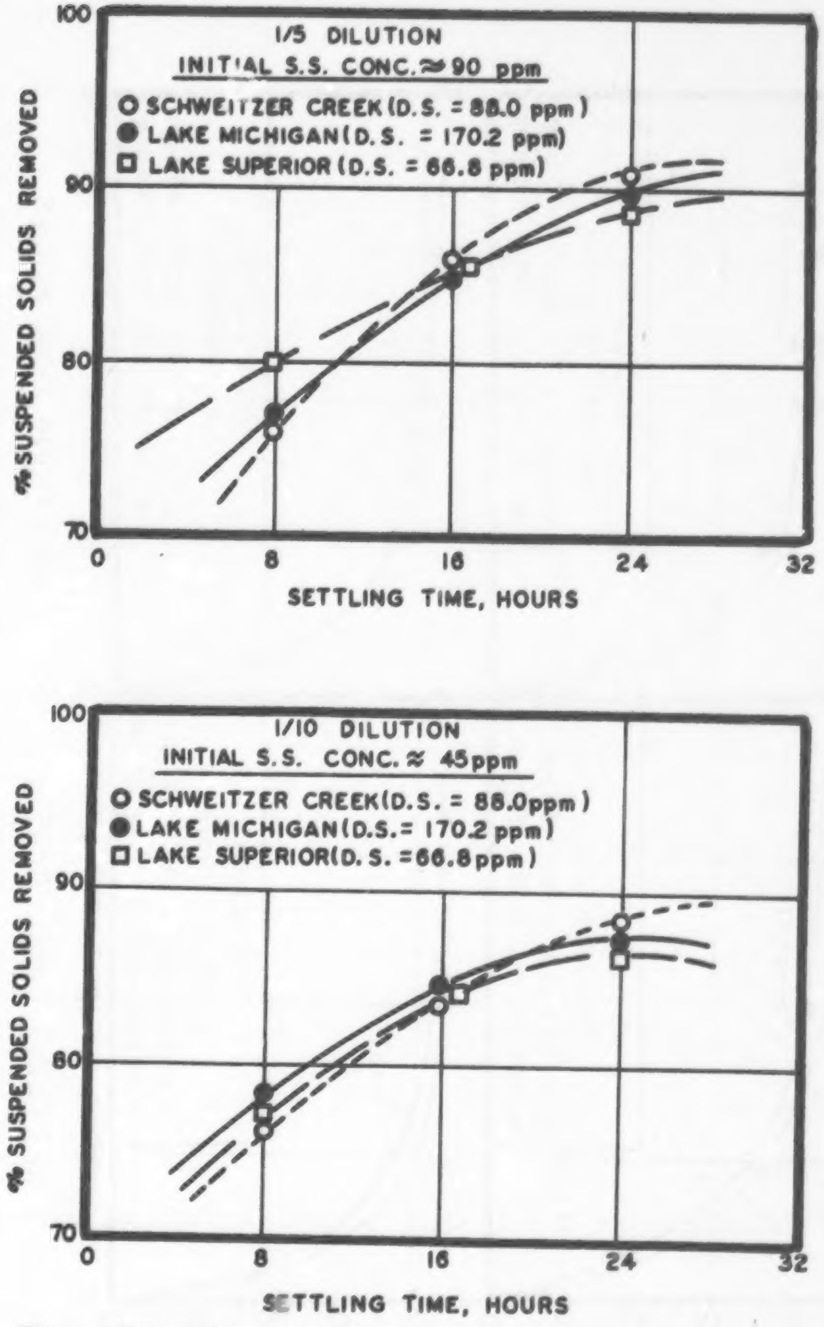


Figure 37. Effect of Various Diluent Waters on the Removal of Empire Fine Tailings at 40° F.

quiescent settling column tests. The column depth of 27 inches approximated the mean depth of the section under study. Suspended solids reductions were predicted from the column data for a time equal to the mean hydraulic detention time of the section. The results of this study, conducted at a flow rate of 12.8 cfs, are shown in Table 20.

Table 20. Results Obtained from Field Study at Republic Mine

Station	3		2		1
Average Suspended Solids Conc., ppm	55.7		55.4		42.0
Percent Suspended Solids Removed in Stream	0		0		24.2
Dentention Time, Hrs.	0		0.5		7.0
Predicted Susp. Solids % Reduction			0		20.0
Average Velocity fps Between Stations		1.1		0.077	

No removal was either obtained or predicted over the section between Stations 3 and 2 because the stream velocity was considerably in excess of the scour velocity for the particles under consideration. The predicted removal over the section between Station 2 and 1 agreed reasonably well with the actual removal.

Particle Size Distributions

Particle size distributions were determined for the Republic and Empire fine tailings by two methods:

1. By applying Stokes' Law to gravity settling data, and
2. By direct measurement of gross dimensions shown on photomicrographs.

According to Stokes' Law:

$$V_s = \frac{d^2}{18\mu}(\gamma_s - \gamma_w) \qquad 11.$$

where: V_s = settling velocity, d = particle diameter, μ = viscosity, γ_s = weight density of solids, γ_w = weight density of the fluid. The mass fraction of particles settling at velocities equal to or less than a given depth divided by a given time was read directly from time settling or iso-percent removal curves e.g. Figures 31 and 32. Then, for known values of u, γ_s and γ_w, Stokes' Law was applied to calculate the diameter of particles settling at the given depth/time. The specific gravity of the Republic and Empire fine tailings particles was determined by standard pycnometer techniques (18) as:

Republic fine tailings: SG = 2.00
Empire fine tailings: SG = 2.63

This method has a few inherent drawbacks when applied to flocculent suspensions exhibiting curvilenear time -settling curves. The particle size distribution determined for these suspensions is affected somewhat by the particular point chosen for analysis on a given time settling curve. In general, choosing points at greater depths and times results in a larger particle size distribution. Likewise, the resulting particle size distribution may be influenced by the characteristics of the diluent water.

The photomicrographs were taken using an American Optical Series 10 Phasestar microscope equipped with a 35 mm camera. The particles shown on the photomicrograph were grouped into 8 size classes and between 15 and 70 particles were counted in each size class.

Figure 38 summarizes the particle size distribution obtained. It can be seen that the diluent water influenced the distribution obtained from gravitational settling data. Generally, the optical method tended to estimate fewer numbers of very small and very large particles. However, the mean particle sizes (50% smaller by weight), determined by each method, agreed very closely for the Republic particles yielding a mean diameter of 1.3 microns. For the Empire particles, the optical method yielded a mean particle size of 6.5 microns whereas the settling data yielded 8.0 microns.

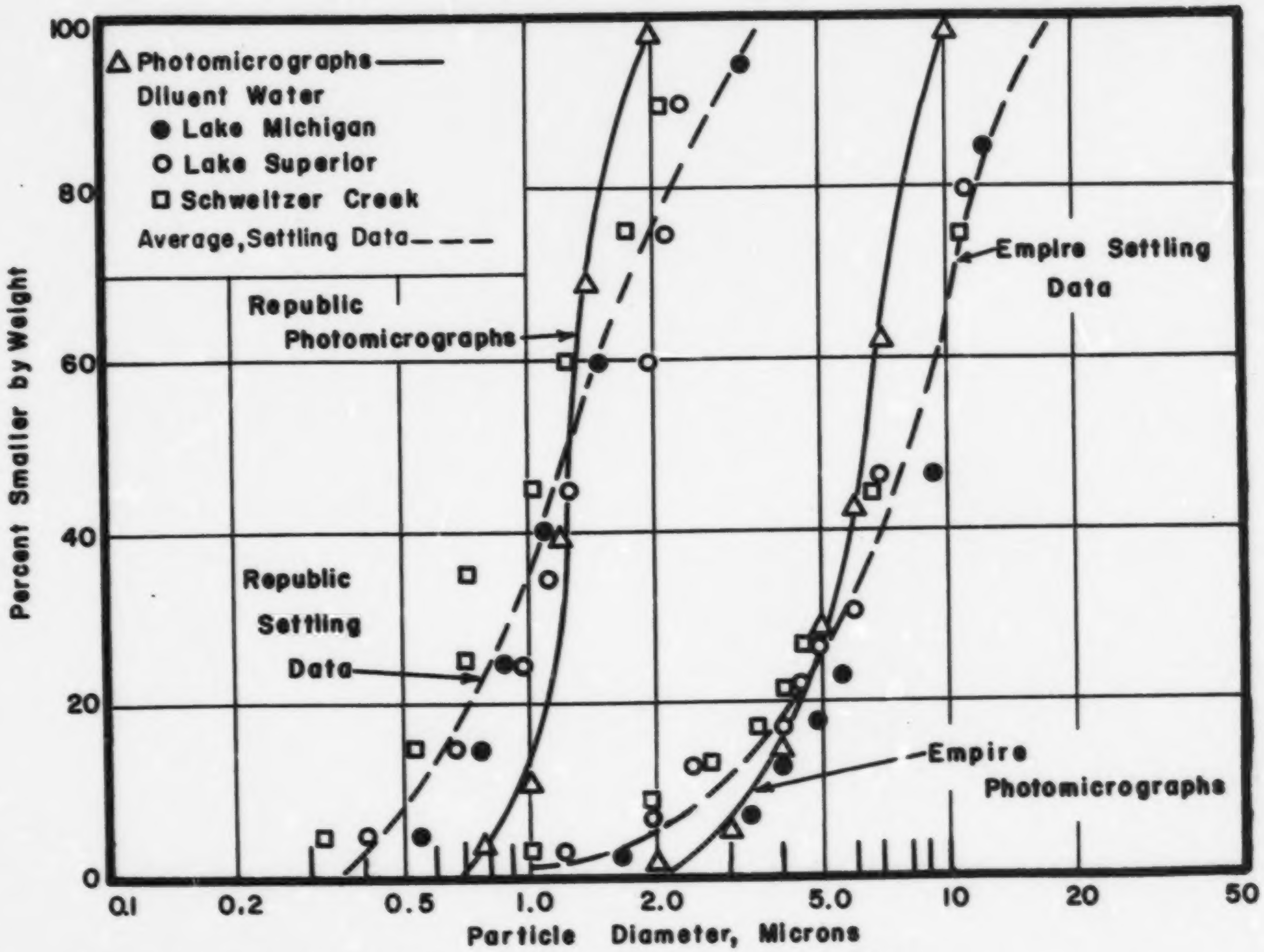


Figure 38. Partical Size Distributions for Fine Tailings.

SECTION VII
ACKNOWLEDGEMENTS

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APPENDIX A

Water Balance and Pond Performance Data

Storage Calculations for ΔS_p and ΔS_t

Republic

The cumulative change in water storage at Republic was determined by considering the changes in water surface levels occurring in the various ponds throughout the year. In 413 acres of the Pond 1 area, the elevations of both the tailings and water surface increased by approximately 4.0 feet during the June 15, 1970 to June 15, 1971 study period. It was determined that the average porosity of the deposit was 46%. This storage was assumed to occur uniformly throughout the period at a rate of:

$$\frac{4\text{ft.}(413\text{ ac})(0.46)}{365\text{ days }\left(1.985\ \frac{\text{ac. ft.}}{\text{sfd}}\right)} = 1.05\text{ cfs}$$

An area of 49.7 acres located at the southern end of Pond 1 (originally part of Pond 2) experienced a water level and tailings rise of 7.2 feet. The average porosity of the tailings in this area was 50%. This storage was assumed to occur uniformly throughout the period at a rate of:

$$\frac{7.2\text{ ft. }(49.7\text{ ac})(0.50)}{365\text{ days }\left(1.985\ \frac{\text{ac. ft.}}{\text{sfd}}\right)} = 0.247\text{ cfs}$$

On April 3, 1971, the main tailings stream was diverted into the Northeast Pond 1 area. This area of 106 acres impounded the entire 58 cfs for a period of 4.35 days until an average water depth of 4.72 feet was attained. At this point water overflowed into the Pond 1 area and near normal conditions were attained. However, the main tailings stream continued to flow into this area of impounded water until June 15, 1971. Thus, the solids were being deposited into water and causing a displacement or negative storage of 1.41 cfs. The net storage in this northeast Pond 1 area over the period of April 3 to June 15, 1971 was, therefore:

$$\frac{106\text{ ac }(4.72\text{ ft.})}{1.985\ \frac{\text{ac.ft.}}{\text{sfd}}} - 1.41\text{ cfs }(72\text{ days}) = 151\text{ sfd}$$

On July 26, 1970, a new set of outlet controls and culverts were put into service at the 3B location. This caused the water surface of Pond 2 to rise. It was estimated that this accounted for 60 sfd of storage over the period of July 26-27.

The water surface elevation of the Return Flow Pond was measured and recorded daily. For Return Flow Pond surface elevations below 1507.15, the water surface elevation of Pond 3 was held constant and by the overflow structure. For higher elevations, the changes in surface elevations of Pond 3 were equal to those of the Return Flow Pond.

The total cumulative change in water storage shown in Column 8 of Table A.2 was determined by adding the cumulative changes calculated for Pond 1, 2, 3 and the Return Flow Pond.

Empire

The cumulative change in water storage at Empire was determined by considering both the changes in water surface elevation of the pond area and the amount of water displaced by the advancing tailings front. The water surface elevation was recorded continuously at the West outlet and the changes in storage caused by changes in the water surface elevation were calculated by multiplying the change in elevation by the average 406 acres of water surface.

The annual tailings accumulation amounted to approximately 2270 acre ft. Field surveys showed that about 80% of this volume was deposited into water as the tailings front advanced. The average porosity of the tailings deposit was determined to be 38%. Thus, the rate at which water was being displaced was calculated as:

$$\frac{0.8\ (2270)\ \text{ac. ft.}\ (0.62)}{1.985\ \frac{\text{ac.ft.}}{\text{sfd}}\ (365\ \text{days})} = 1.55\ \text{cfs}$$

This was considered as a negative storage.

Because the pond water surface rose by nearly 2 feet during the study period, most of the remaining 20% (which was not deposited directly into the water) resided at a position below that of the final water surface. Thus, the rate at which water was retained in this fraction was estimated as:

$$\frac{0.2\ (2270)\ \text{ac. ft.}\ (0.38)}{1.985\ \frac{\text{ac.ft.}}{\text{sfd}}\ \ 365\ \text{days}} = 0.24\ \text{cfs}$$

Thus, the net uniform storage was

$$0.24 \text{ cfs} - 1.55 \text{ cfs} = -1.31 \text{ cfs}$$

or, 1.31 cfs was continuously released by displacement.

The total cumulative change in water storage shown in column 6 of Table A.1 was determined by subtracting the cumulative amount of water released by displacement from the cumulative volume added by the rise in the water surface.

Seepage Estimate

Republic

Applying Equation 7 over the period of June 15 to October 31, 1970 gives

$$\frac{L}{t} = \frac{(I+P-E) - (\Delta S_p + \Delta S_t + Q)}{t}$$

$$\frac{L}{t} = \frac{8257 \text{ sfd} - 7454 \text{ sfd}}{139 \text{ days}} = 5.77 \text{ cfs}$$

The seepage over the pond of June 15, 1970 to March 31, 1971 is

$$L = 5.77 \text{ cfs } (290 \text{ days}) = 1672 \text{ sfd}$$

Since the total seepage was 2463 sfd, the seepage rate over the period of April 1 to June 28, 1971 is given by

$$\frac{2463 \text{ sfd} - 1672 \text{ sfd}}{89 \text{ days}} = 8.9 \text{ cfs}$$

Empire

Applying Equation 7 over the period of July 8, 1970 to June 28, 1971 gives

$$\frac{L}{t} = \frac{4715 \text{ sfd} - 1962 \text{ sfd}}{355 \text{ days}} = 7.75 \text{ cfs}$$

Table A.1. Empire Tailings Area - Water Balance Data

	NET INPUT sfd				OUTFLOW sfd	
(0) Date	(1) Plant Inflow I	(2) Precip. P	(3) Evap. E	(4) Cumul. Total $\Sigma(I+P-E)$	(5) Cumul. Total ΣQ	(6) Cumulative Change In Water Storage $\Sigma(\Delta S_p + \Delta S_t)$
7/8/70					8	0
7/10					24	-11
7/20					89	-50
7/31	218	356	338	236	137	-78
8/10					174	-91
8/20					220	-103
8/31	324	81	137	504	290	-68
9/10					374	-94
9/20					447	-107
9/30	306	470	223	1057	517	-95
10/10					562	-30
10/20					598	-31
10/31	303	495	137	1718	660	+2
11/10					702	-11
11/20					733	-3
11/30	289	230	95	2142	777	-16
12/10					828	-38
12/20					868	-60
12/31	299	150	43	2548	908	-63
1/10/71					950	-103
1/20					977	-54
1/31	305	306	0	3159	1022	-107
2/10					1062	-121
2/20					1103	-134
2/28	294	218	86	3585	1135	-145
3/10					1169	-159
3/20					1205	-172
3/31	303	177	178	3877	1245	-220
4/10					1308	-46
4/20					1510	-36
4/30	270	121	148	4130	1645	-60
5/10					1755	-122
5/20					1848	-116
5/31	327	217	240	4434	1959	-147
6/10					2046	-177
6/20					2125	-159
6/28	292	353	364	4715	2182	-220

Table A.1. Empire Tailings Area - Water Balance Data (Continued)

(0) Date	(7) Cumul. Tot. Outflow + Storage sfd (5) + (6) $\Sigma(Q+\Delta S_p+\Delta S_t)$	(8) Cumul. Total Seepage sfd ΣL	(9) Cumul. Tot. Input Less Seepage sfd (4) - (8) $\Sigma(I+P-E-L)$	(10) Unaccounted Storage sfd $\Sigma(\Delta S_s+\Delta S_l+\Delta S_g)$	(11) Snow Storage on Watershed & Tailings sfd $\Sigma(\Delta S_s)$
7/8/70	8				
7/10	13				
7/20	39				
7/31	59	186	50	-9	0
8/10	83				
8/20	117				
8/31	222	426	78	-144	0
9/10	280				
9/20	340				
9/30	422	658	399	-23	0
10/10	532				
10/20	567				
10/31	662	899	819	+157	0
11/10	691				
11/20	730				
11/30	761	1130	1012	+251	68
12/10	790				
12/20	808				
12/31	845	1372	1176	+331	212
1/10/71	847				
1/20	923				
1/31	915	1612	1547	+632	466
2/10	941				
2/20	969				
2/28	990	1828	1757	+767	480
3/10	1010				
3/20	1033				
3/31	1025	2070	1817	+792	480
4/10	1262				
4/20	1474				
4/30	1585	2300	1830	+245	0
5/10	1633				
5/20	1732				
5/31	1812	2540	1894	+82	0
6/10	1869				
6/20	1966				
6/28	1962	2753	1962	0	0

Table A.2. Republic Tailings Area - Water Balance Data

(0) Dates	NET INPUT, sfd (1) Plant Inflow I	(2) Precip. P	(3) Evap. E	(4) Cumul. Total Σ(I+P-E)	CUMULATIVE OUTPUT, sfd (5) 5-B Outlet	(6) Return Flow	(7) Total Output ΣQ
6/15/70					1.6	50.1	51.7
6/20					8.4	291.3	299.7
6/30	926	31	65	892	16.4	777.3	793.7
7/10					78.9	1141.5	1220.4
7/20					186.0	1521.6	1707.6
7/31	1800	212	201	2703	307.4	1958.1	2265.5
8/10					493.5	2330.4	2823.9
8/20					633.1	2739.9	3373.0
8/31	1800	37	102	4438	759.6	3165.9	3925.5
9/10					938.6	3530.7	4469.3
9/20					1102.9	3916.2	5019.1
9/30	1740	216	102	6292	1316.2	4271.7	5587.9
10/10					1445.3	4636.2	6081.5
10/20					1589.7	4987.5	6577.2
10/31	1800	228	63	8257	1833.6	5344.2	7177.8
11/10					2022.5	5710.8	7733.3
11/20					2105.6	6195.0	8300.6
11/30	1740	106	43	10060	2201.5	6570.9	8772.4
12/10					2340.6	6956.1	9296.7
12/20					2457.7	7372.8	9830.5
12/31	1800	69	20	11909	2496.3	7856.4	10352.7
1/10/71					2569.9	8244.9	10814.8
1/20					2646.1	8648.7	11294.8
1/31	1800	141	0	13850	2708.6	9039.3	11747.9
2/10					2711.6	9497.4	12209,0
2/20					2737.2	9856.8	12594.0
2/28	1625	100	39	15536	2859.8	10140.0	12999.8
3/10					3067.3	10417.2	13484.5
3/20					3264.5	10748.4	14012.9
3/31	1800	80	80	17336	3542.9	11148.3	14691.2
4/10					3759.3	11495.5	15254.8
4/20					4086.8	11794.5	15881.3
4/30	1740	56	87	19045	4354.2	12003.0	16357.2
5/10					4600.9	12217.2	16818.1
5/20					4726.0	12574.2	17300.2
5/31	1800	100	127	20818	4909.0	12888.0	17797.0
6/10					5105.9	13177.8	18383.7
6/20					5296.0	13491.9	18787.9
6/28	1625	151	163	22431	5441.2	13779.0	19220.2

Table A.2. Republic Tailings Area - Water Balance Data (Continued)

	CUMULATIVE, sfd					
(0) Dates	(8) Change In Water Storage $\Sigma(\Delta S_p + \Delta S_t)$	(9) Total Output +Storage $\Sigma(Q + \Delta S_p + \Delta S_t)$	(10) Total Seepage ΣL	(11) Total Input -Seepage (4)-(10) $\Sigma(I+P-E-L)$	(12) Unaccounted Storage (11)-(9) sfd $\Sigma(\Delta S_s + \Delta S_l + \Delta S_g)$	(13) Snow Storage sfd $\Sigma(\Delta S_s)$
6/15/70	0	51.7				
6/20	12.1	311.8				
6/30	1.7	795.4	92	800	+ 4.6	0
7/10	71.8	1292.2				
7/20	95.3	1802.9				
7/31	164.8	2430.3	271	2432	+ 1.7	0
8/10	140.9	2964.8				
8/20	132.1	3505.1				
8/31	155.1	4080.6	450	3988	- 93	0
9/10	182.4	4651.7				
9/20	186.1	5202.2				
9/30	204.7	5792.6	623	5669	-124	0
10/10	227.7	6309.2				
10/20	220.1	6797.3				
10/31	276.1	7453.9	803	7454	0	0
11/10	270.4	8003.7				
11/20	255.4	8556.0				
11/30	304.6	9077.0	975	9085	+ 8	38
12/10	302.0	9598.7				
12/20	315.0	10145.5				
12/31	327.5	10680.2	1155	10754	+ 74	81
1/10/71	347.3	11162.1				
1/20	334.7	11629.5				
1/31	334.7	12082.6	1335	12515	+432	86
2/10	329.0	12538.0				
2/20	447.3	13041.3				
2/28	434.0	13433.8	1495	14041	+607	73
3/10	392.6	13877.1				
3/20	427.0	14439.9				
3/31	422.2	15113.4	1675	15661	+548	60
4/10	742.4	15997.2				
4/20	712.8	16594.1				
4/30	705.5	17062.7	1942	17103	+ 41	0
5/10	695.2	17513.3				
5/20	730.4	18030.6				
5/31	729.2	18526.2	2217	18601	+ 75	0
6/10	738.8	19022.5				
6/20	742.4	19530.3				
6/28	747.7	19967.9	2463	19968	0	0

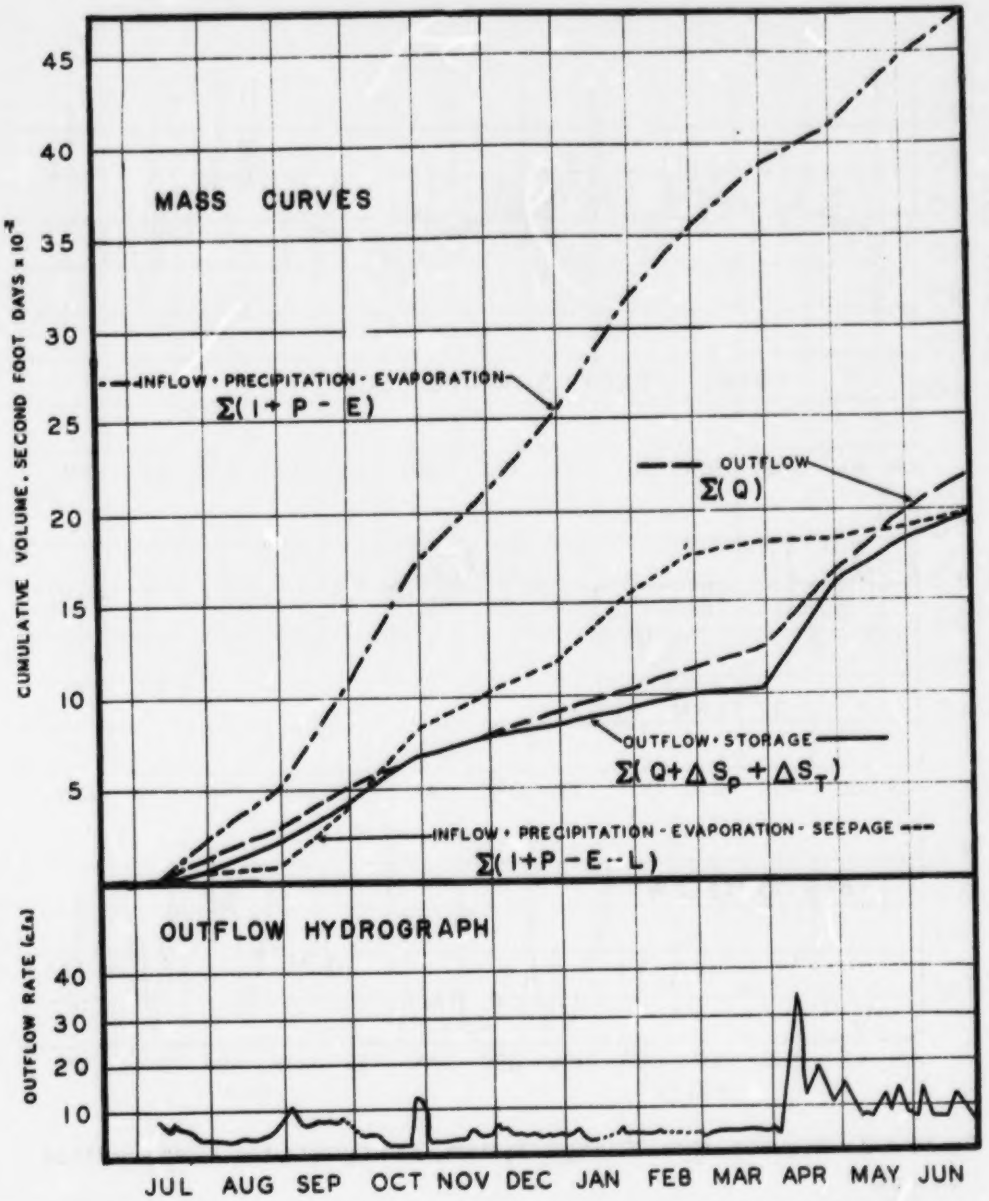

Figure A.1. Empire Tailings System, Outflow Hydrograph and Mass Curves.

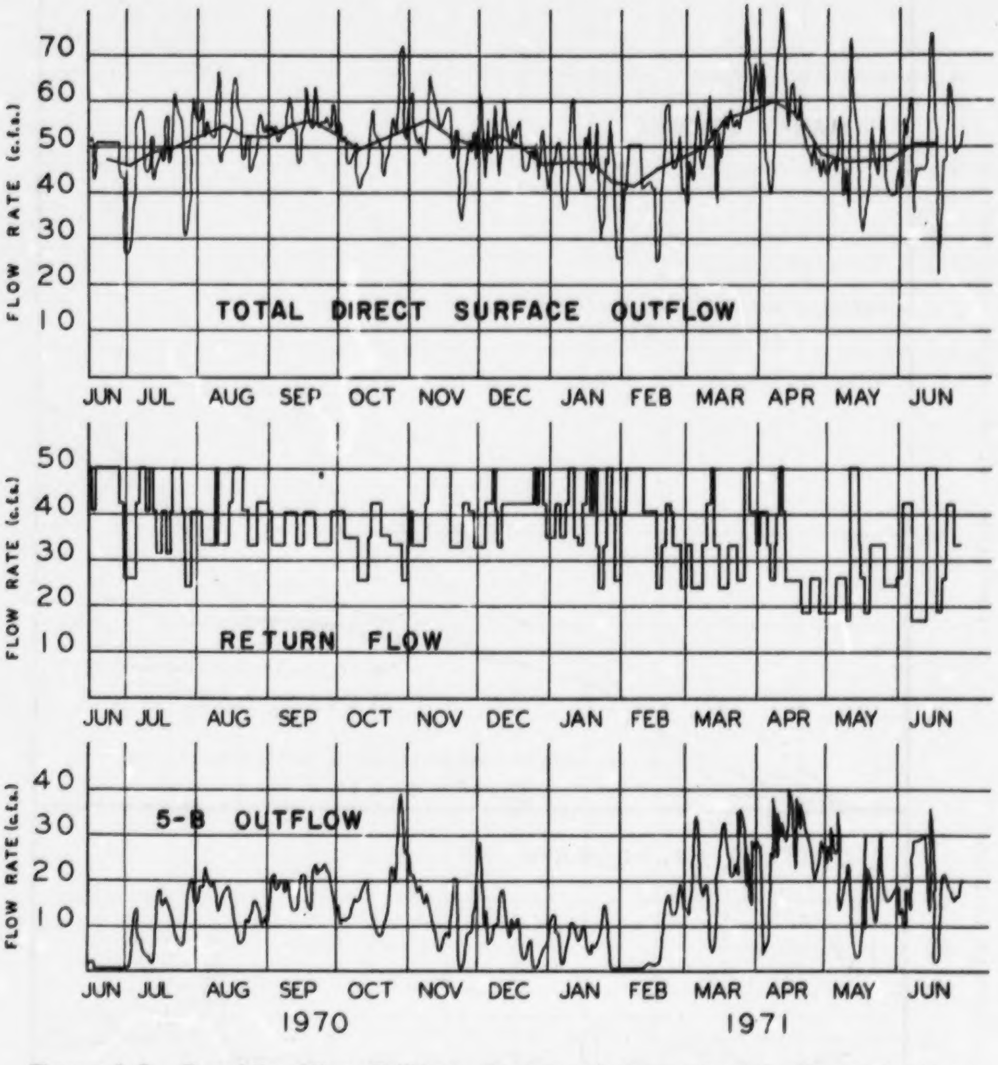

Figure A.2. Republic Mine Tailings System, Hydrographs for Direct Surface Outflow.

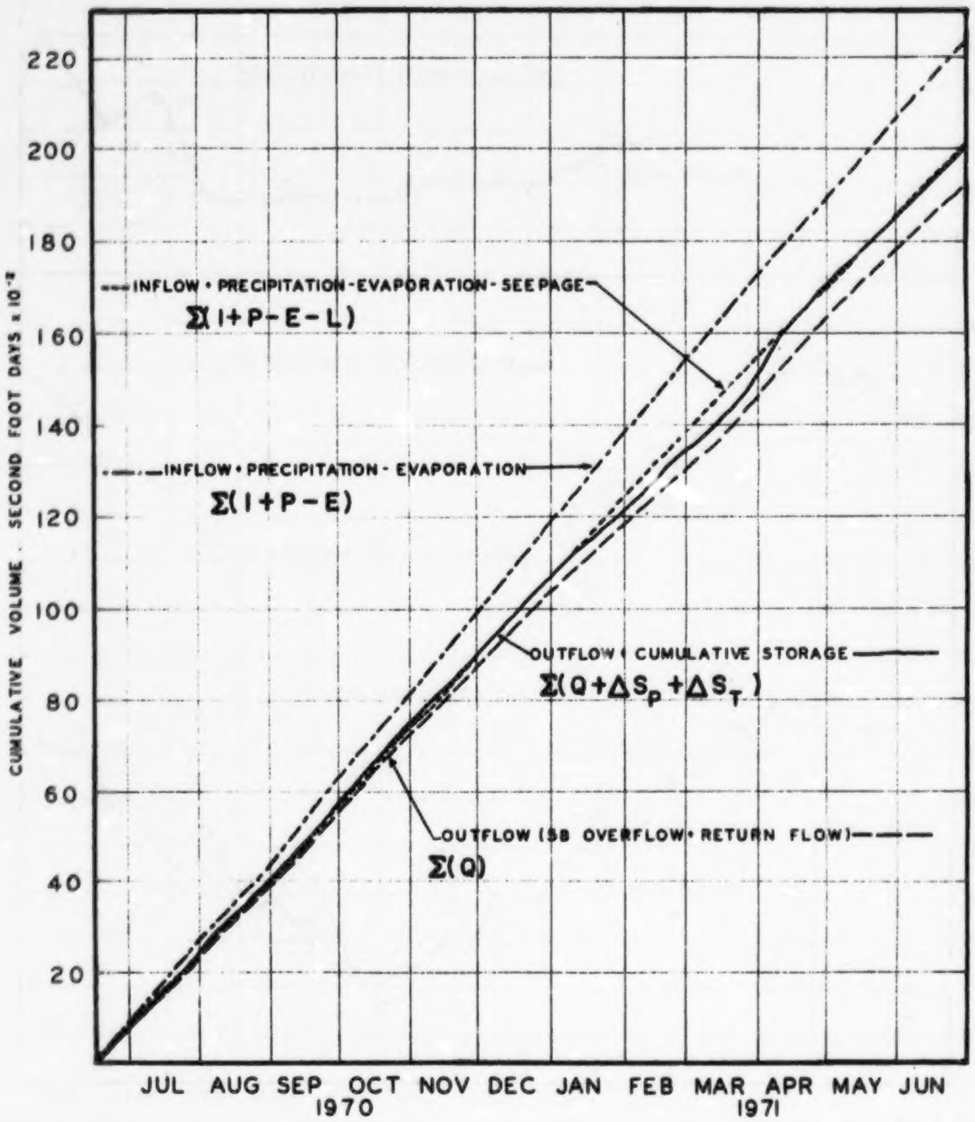

Figure A.3. Republic Tailings System, Mass Curves.

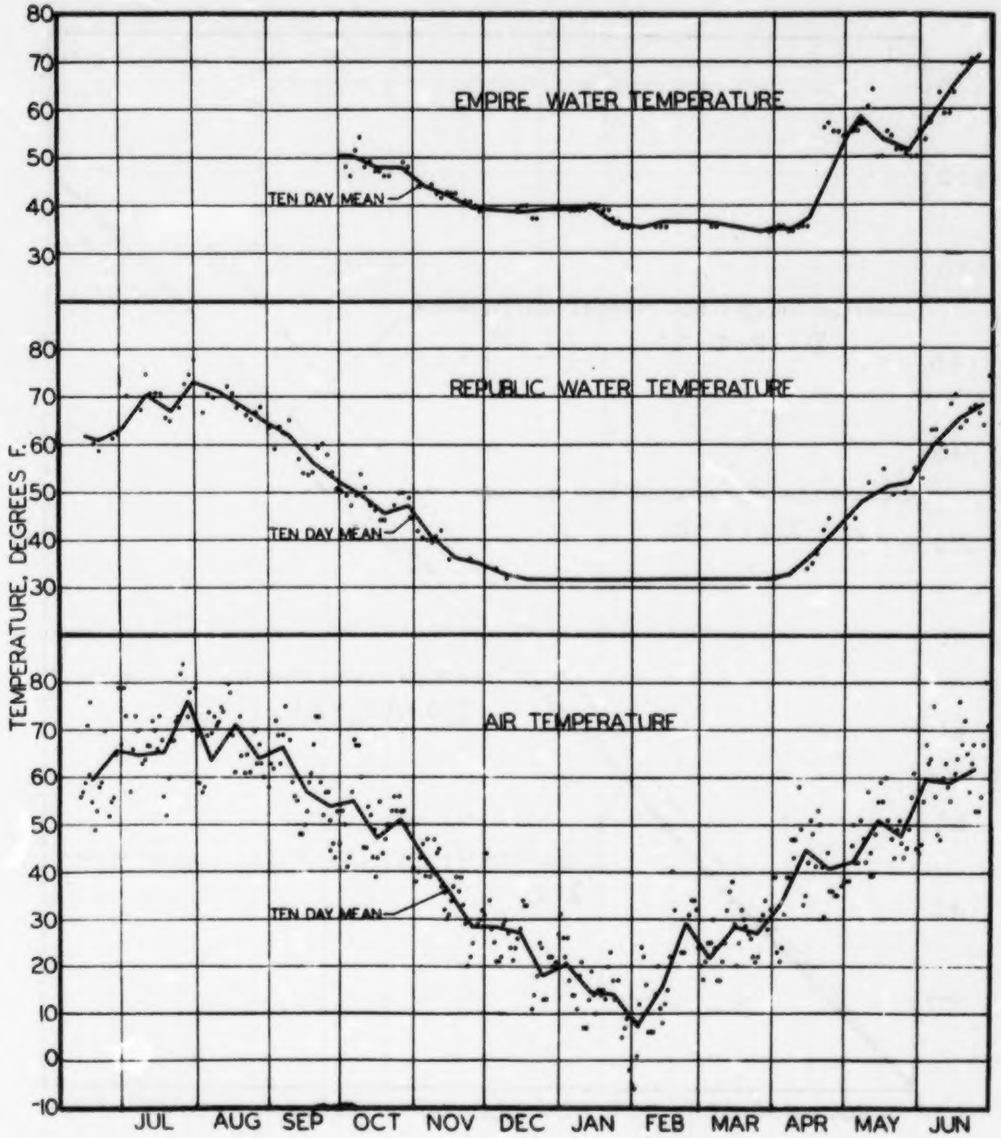

Figure A.4. Variation in Water and Air Temperatures for the Study Sites.

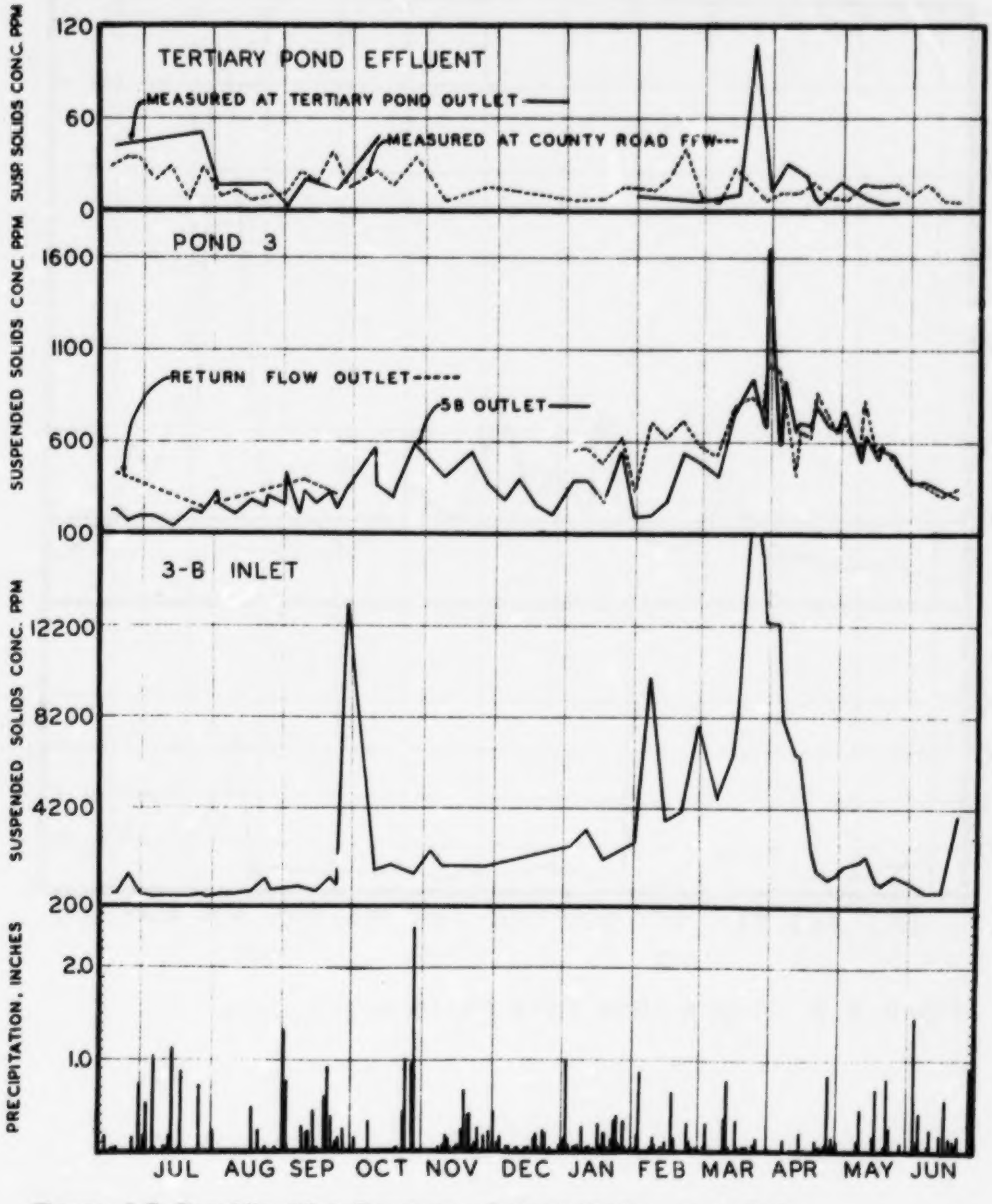

Figure A.5. Republic Mine, Summary of Pond Performance Data.

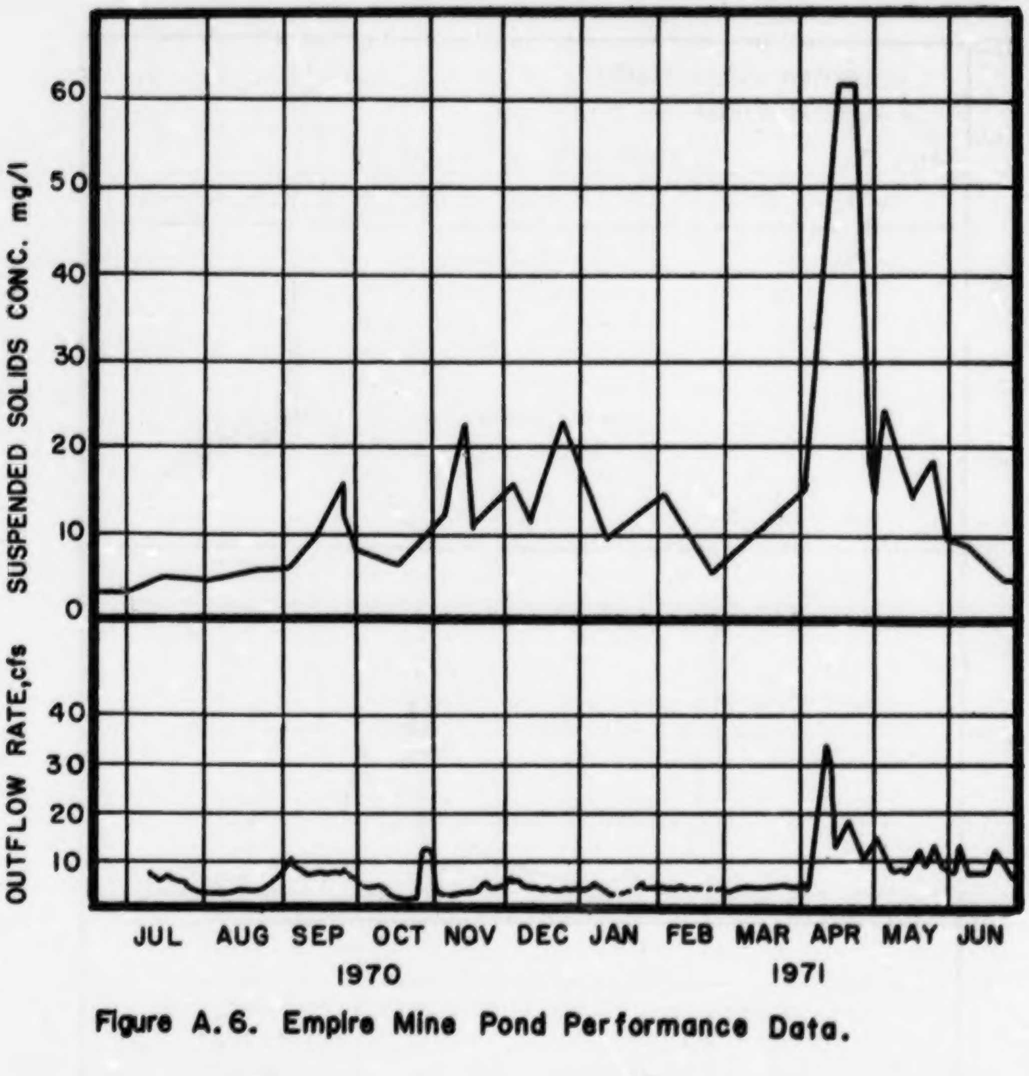

Figure A.6. Empire Mine Pond Performance Data.

APPENDIX B

Quiescent Settling Tests: Methodology

Samples of the tailings waste used for quiescent settling tests were collected in 5 gallon polyethylene containers from the main waste flow immediately after leaving the mills. The following procedure was used to investigate the settling characteristics of the tailings material.

(1) A representative sample of the main tailings waste was stored in the collecting container until the suspension reached its test temperature.

(2) After being thoroughly mixed, the sample was presettled for various time periods depending upon the desired initial suspended solids concentration.

(3) The supernatant from the presettled suspension was poured into a 4 gallon bucket and mixed before being introduced into the column.

(4) Compressed air was then supplied through the valved outlet in the bottom of the column to create a uniform suspended solids concentration over the entire length of the column.

(5) Immediately after the application of compressed air a horizontal screen was dragged slowly upward from the bottom to maintain uniform particle distribution and to damp out any turbulence and recirculation eddies.

(6) Settling time was started when the screen reached the top.

(7) Immediately after starting the settling time a sample was withdrawn from the top, middle and bottom sample port to bring the surface level to a predetermined distance from the bottom. The collected samples were used to determine initial suspended solids concentration, dissolved solids concentration, pH-value and to establish the uniformity of the suspended solids concentration.

(8) The suspension was then allowed to settle under quiescent conditions.

(9) At each time interval, column samples (60 ml) were extracted by use of a syringe and a specially designed sampling port permitting the extraction of a representative sample from an undisturbed cross-sectional area at different depths with negligible leakage.

(10) The suspended solids concentration of each sample was determined using the membrane filtration technique of Winneberger et. al. (19). A Mettler, H-20, analytical balance was employed in the determination of the suspended solids concentrations. The dissolved solids concentrations were determined in accordance with the procedure outlined in Standard Methods (20). A Corning, Model 12, research pH meter was used to measure the pH-values of the sample suspension.

A list of the quiescent settling tests performed is given in Table B.1. The test temperatures were maintained at ± 1°F by placing the settling column with its suspension in a closed room where near constant temperature conditions were maintained. The given temperatures were chosen because the clarification ponds at Republic Mine was found to have a maximum temperature of approximately 75°F during the summer and the fact that water reaches its maximum density at 39.2°F.

To determine the effect of addition of a coagulant material and in order to predict the performance efficiency of the Tertiary Pond, certain settling tests were performed with the addition of 10 to 25 ppm Alum and 2 ppm Polymer (Nalco 603-Cationic). The above dosages are typical of those used at Republic Mine to treat the discharge from Pond 3. When coagulants were added, the following column test procedure was followed.

(1) After filling the columns with a representative sample of fine tailings from Pond 3 overflow the suspension was mixed by compressed air and the surface was brought to a predetermined level by withdrawing samples from top, middle, and bottom sampling port. The samples were analyzed for initial suspended solids concentration, dissolved concentration, and pH-value.

(2) A predetermined flow of air was then supplied through the valved outlet at bottom of column and the exact dosage of alum was injected by a syringe through the bottom sampling port.

(3) The suspension was mixed rapidly and then slowly by compressed air for a certain period of time after which a horizontal screen was used to damp out any turbulence and the settling time started.

Table B.1. Summary of Quiescent Settling Test Conditions - Republic Tailings

Run No.	Column No.	Presettling Time	Alum mg/l	Polymer mg/l	Initial Fluid Depth Inches	Initial S.S. mg/l	Dissolved Solids mg/l	Average Fluid Temp. °F	pH
1	1	2 hrs.	0	0	72	3,450	150	75	7.8
	2	2 hrs.	10	0	72	3,450	150	75	7.8
2	3	15 min.	0	0	69	7,733	--	41	--
	4	15 min.	0	0	69	7,957	--	41	--
3	1	15 min.	0	0	71	8,195	--	39	--
	3	15 min.	0	0	71	8,014	--	72.5	--
4	3	--	0	0	71	7,298	364	72	--
5	1	20 min.	0	0	71	11,480	134	39	--
	3	20 min.	0	0	71	9,729	134	74	--
6	1	20 min.	0	0	71	12,674	130	39	--
	3	20 min.	0	0	71	12,447	130	72	--
7	1	0	0	0	71	425	114	41	7.1
	2	0	25	0	71	303	114	41.5	7.1
	3	0	0	0	70.5	400	114	73	7.1
	4	0	25	0	70	402	114	73	7.1
8	1	0	25	2	71	293	120	41	7.2
	3	0	25	0	71	265	117	75	7.2
	4	0	25	2	70	451	120	75	7.5

(4) If alum and polymer were added, the polymer was added with a syringe through the bottom sampling port after a short period of rapid mixing with alum.

(5) The suspension was continuously rapid mixed during injection of the polymer and for a certain time period thereafter.

(6) After a period of slow mixing the turbulence was damped and the settling time started.

A list of the mixing times and the amount of air used for the various tests are given in Table B.2. The data in Table B.2. are based on equations for mean velocity gradient as given by Fair and Geyer (9).

Table B.2. Mixing Conditions Used for Addition of Alum and Polymer

		Rapid Mixing			Slow Mixing		
Coagulant	Test Temp. ^{o}F	Time min.	Velocity Gradient ft/sec/ft	Air Flow cm^3/min	Time min.	Velocity Gradient ft/sec/ft	Air Flow cm^3/min
Alum	75	7	153	943	17	47	106
	40	7	113	1035	17	34.9	98.5
Alum and Polymer	75	4+3	153	943	17	47	106
	40	4+3	113	1035	17	34.9	98.5

Settling Columns. The quiescent settling tests were performed in 4" I.D. columns made of 1/4" thick transparent acrylic resin tubing approximately 74" long and mounted on 1/2" thick acrylic resin base plates. Sample ports were located at 12" intervals.

The average depth, defined as Volume/Area, of the Republic Mine clarification basins was determined to be approximately 57". Thus a total column length of 74" with an effective settling depth of maximum 66" was used.

Effective Settling Distance. The water surface drop caused by extracting samples from the suspension at different time intervals changed the settling distance for each sampling run. This decrease in the settling distance was accounted for when making iso-concentration plots in terms of depth and time. A sampling volume of 60 ml was used for all tests. This resulted in a surface level drop of approximately 5/16 inch for each sample extracted.

APPENDIX C

Dye Tracer Methodology

At Republic's Pond 3 and at the Empire Pond the tracer was injected into a channel which had formed near the tailings front. The injection was accomplished by using a 5 gallon polyethylene container with a 3 inch valved pipe connected to the container mouth. The tracer was injected in less than 10 seconds at a depth equal to 60% of the total depth.

Continuous sampling and fluorescence measurement were employed for the relatively short term (less than 70 hours) tests conducted on the Tertiary Pond. However, for the larger ponds, the tests covered periods ranging from 250 hours to 350 hours. Since it was impractical to maintain the portable alternator and other equipment associated with the continuous sampling and flourescence measurement over this time period, individual samples were collected and analyzed in the laboratory.

Of the many different fluorescent dyes available, rhodamine B and rhodamine WT were chosen for this study. Rhodamine WT was chosen because of its relatively low adsorption on the tailings fine particles and its high fluorescent strength. Rhodamine B was chosen because of its low cost and high fluorescent strength. It has been established that rhodamine B exhibits a moderate sorptive tendency (21). Rhodamine B was used to determine the hydraulic characteristics of both the Tertiary Pond and Empire Pond which had relatively low concentrations of suspended solids. Rhodamine WT was used for Pond 3 which had a high suspended solids concentration throughout its entire volume.

Rhodamine WT is a relatively new fluorescent tracer and little information is available in the literature on its use. The manufacturer (du Pont) describes rhodamine WT as a dark red 20 percent solution with a specific gravity of 1.19 + 0.02 at 20°C and a low tendency to stain silt, dirt and other suspended matter in shallow and inland waters. The effect of temperature on the fluorescent intensity of rhodamine WT has been established by Wilson (21). To eliminate the effect of the temperature all rhodamine WT analyses were performed at the same temperature. No information on the photochemical decay rate of rhodamine WT could be found, but it is believed to be similar to that of rhodamine B (21). The effect of photochemical decay was negligible since little light penetrated the Pond 3 suspension because of the high solids concentration. Because the specific gravity of the 20% solution or rhodamine WT (1.19) differed from that of water, methyl alcohol was added to the dye solution to adjust the specific gravity to approximately 1.00. This promoted effective mixing between the water and tracer so that the tracer dispersed uniformly throughout the cross section of the influent stream before entering the pond.

Rhodamine B is available in powder and solution form. The powder form was used for this study. The behavior of rhodamine B as a tracer material has been investigated by Feuerstein et. al. (22). They found the fluorescent intensity of rhodamine B to vary markedly with sample temperature, but the photochemical decay rate was found to be low. Temperature correction curves constructed by Wilson (21) were employed in determining the actual concentrations of rhodamine B.

The determination of fluorescent intensities was made with a G.K. Turner Associates Model 111 Fluorometer. A far UV lamp was used as light source. Filter 546 was used as a primary filter and filter 590 was used as a secondary filter for the analyses of both rhodamine B and WT. For high fluorescent intensities a 10 percent neutral-density (ND) filter was placed over the secondary filter. The fluorometer was equipped with a high volume continuous-flow-through door. A sample inlet device consisting of a glass funnel and 1/4" tygon tubing was employed to measure fluorescent intensities of individual samples under conditons of constant temperature. Before the fluorescent intensities in the samples from Pond 3 could be analyzed, the samples were filtered using Millipore Filters (0.45 micron pore size). After adjusting the filtrate to the standard test temperature, 72°F, the fluorescent intensity was determined by continuously adding the filtrate to the sample inlet device. In this way, the sample in the flow cell was continuously replaced to prevent any increase in the sample temperature caused by heat from the light source. All fluorescence measurements were corrected for natural background fluorescence.

APPENDIX D

Prediction of Effluent Concentration by Use of Settling and Dispersion Curves

The method employed to predict basin performance by combining residence time distribution and settling curves is based on theory described by Levenspiel (23), Kramers et. al. (24), and Baillod and Christensen (26). In this analysis, vertical mixing is neglected and fluid elements are assumed to travel through the basin as small, intact columns.

The dye dispersion curve consists of a plot of c versus t. The residence time distribution curve consists of a plot of relative tracer concentration, $E = C/C_0$, versus relative time, $\tau = t/\theta$. The time θ is determined from the dispersion curve by

$$\theta = \frac{\Sigma tc}{\Sigma c}$$

and C_0 is also determined from the dispersion curve by

$$C_0 = \frac{\int_0^\infty c\,dt}{\theta}$$

The performance curve for an ideal plug flow basin consists of a plot of C_{col} versus τ . C_{col} is determined from quiescent settling data for various values of t. Finally, the predicted effluent concentration is calculated by graphical integration as shown in Figure D.1.

$$\begin{array}{c}\text{Predicted}\\\text{Effluent}\\\text{Concentration}\end{array} = \int_0^\infty C_{col}\,E\,d\tau$$

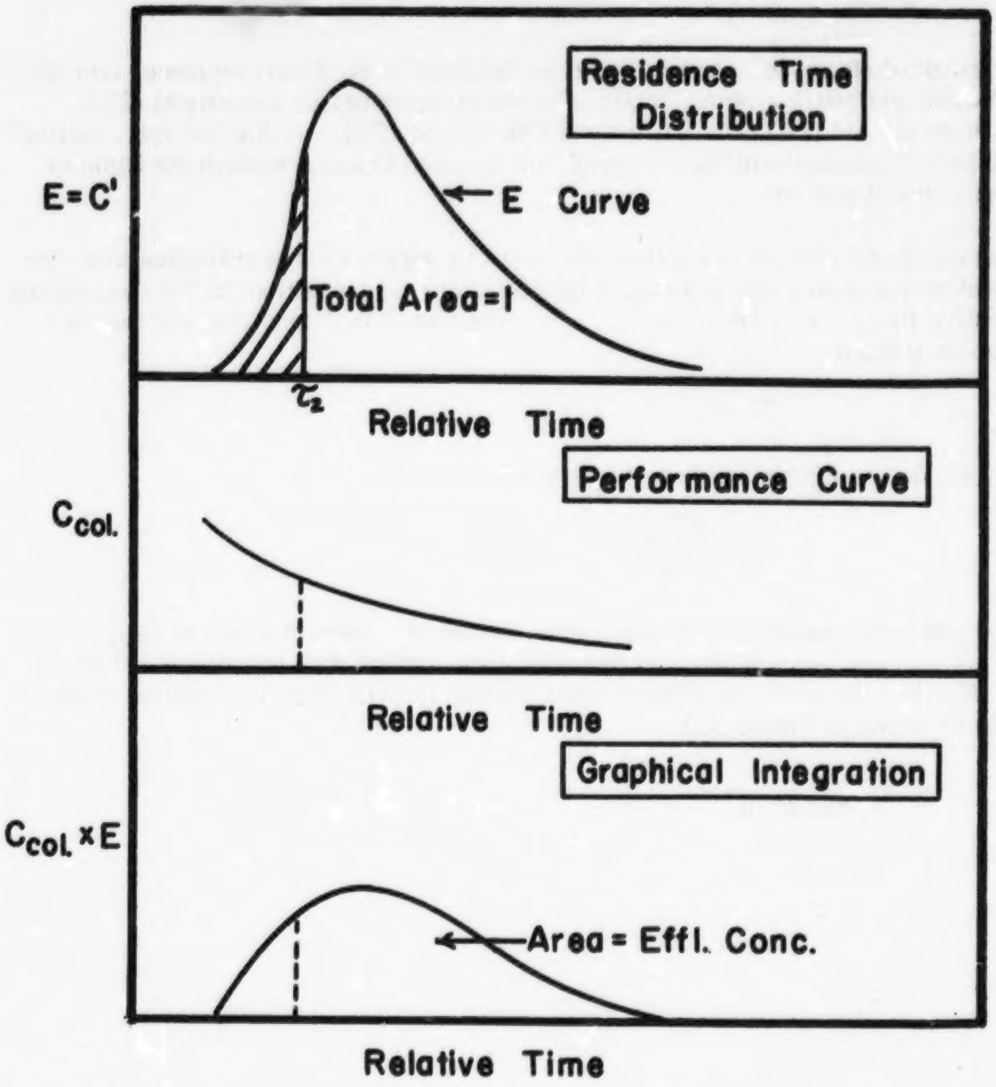

Figure D.1. Prediction of Effluent Concentration by Combining Residence Time Distribution and Performance Curves.

APPENDIX E

Methodology Employed in Coagulent Evaluations

Equipment, Materials, and Analyses. Equipment used, and analyses performed throughout the research included: a Corning Model 12 Research pH Meter, used in sample pH and total alkalinity measurement, a Hach Model 1860-A Turbidmeter, used to measure sample turbidity, Phipps-Bird multiple stirring machine, used in 'jar testing' with the various coagulants and coagulant aids; and standard Imhoff cones, used to measure volumes of sludge settled in 30 minutes (following coagulation-flocculation). The alum used in the research was a reagent grade of hydrated aluminum sulfate. Anionic, cationic, and non-ionic polymer coagulants or coagulant aids were donated by the Nalco Chemical Company, the Calgon Corporation, and the Dow Chemical Company.

Sample Collection and Preparation. Samples that were used in the laboratory coagulation-flocculation tests were collected at the 5-B outlet, located immediately upstream of the alum addition point. Figure E.1. illustrates the relationship between sample turbidity and suspended solids concentration.

Coagulant Preparation. Standard solutions of polymer coagulants were prepared according to manufacturers' instructions. Alum and polymers were added as solutions of 1 and 0.1 percent, respectively, to the several water samples tested in a series of jar tests.

Standard Testing Procedures. The jar test was selected as the primary method for the evaluation of the coagulation-flocculation-quiescent settling process. The procedure was a combination of several practices, largely being patterned after Cohen (25). Raw sample pH levels were adjusted to a predetermined initial pH value by the addition of hydrochloric acid (HCl) or sodium hydroxide (NaOH), and the samples were placed into 1 liter beakers. The beakers were then placed in a plexiglass water bath, and ice or tap water was added to the bath. Circulation of the tank contents by air-induced mixing, and further additions of ice or tap water aided in maintaining water bath and sample temperatures at a specified temperature. Temperatures of 1, 13, and 25^{o} Centigrade were used in laboratory jar testing procedures. At no time did measured temperatures deviate from specified levels by more than $\pm$ 0.5^{o}C.

Coagulants were added to the series of beakers simultaneously, by the use of test tubes mounted in a plastic bar. These were turned upside down, with the tip protruding just underneath the water surface as the coagulant was added to the several beakers. Alum was added at the commencement of rapid mixing, whereas polymers were added 4 minutes after the start of rapid mixing.

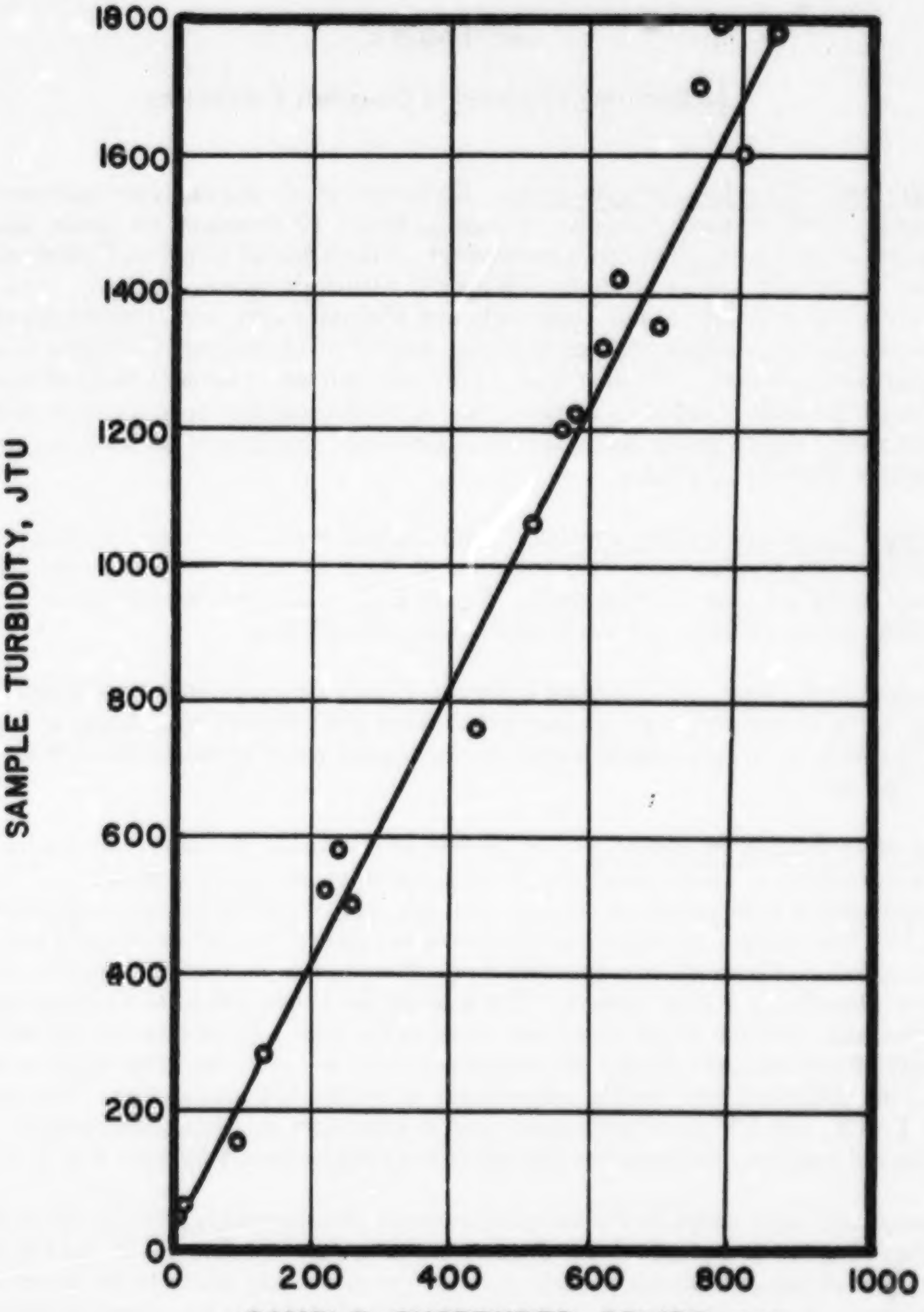

Figure E.1. Correlation between Turbidity and Suspended Solids Concencentration for Republic Pond 3 Overflow.

Supernatant turbidity samples were simultaneously withdrawn by the use of vacuum lines. The withdrawal point was located at mid-depth of a beaker. Unless otherwise specified, a standard quiescent settling time of 30 minutes elapsed between the termination of slow mixing and the measurement of residual turbidity.

Residual pH and residual alkalinity values of the supernatant samples were usually measured, after quiescent settling. Following quiescent settling, beaker contents were stirred to homogenity with a minimum amount of hand stirring. Beaker contents were then gently poured into Imhoff cones, where they were allowed to settle for a period of 30 minutes. Comparisons of sludge volumes formed by various coagulants were possible with this procedure.

Rapid Mixing Operations. In order to facilitate comparisons between coagulation process efficiencies predicted in the laboratory and those attained in the field, rapid mixing conditions were selected to approximate field conditions at the Republic Mine. A rapid mixing speed of 90 rpm (machine maximum = 100 rpm) was selected. It was originally estimated that the effective rapid mix time obtained under field conditions was about 7 minutes. Residence time studies conducted later indicated that the actual rapid mix time was on the order of 4 minutes. Nevertheless, 7 minutes was employed for rapid mixing in all experiments. Thus, the standard rapid mix conditions were 7 minutes at 90 rpm. At 13°F, this resulted in a temporal mean velocity gradient, $G = 135$ fps/ft, and $GT = 5.67 \times 10^4$. The value of G was somewhat less than that normally employed in rapid mix design whereas the value of GT was somewhat greater than that normally employed (10).

Slow Mixing Conditions. In order to determine the optimal slow mixing conditions for a given temperature, a series of experiments was conducted to evaluate the effects of various levels of slow mixing and temperature upon turbidity removals at an initial pH of 7.0 and 50 ppm of alum. Slow mixing speeds of 30, 40 and 50 rpm and temperatures of 1°C, 13°C and 25°C were employed. The values of the temporal mean velocity gradient, G, were determined using data published by Camp (11). Mixing time and temperature were varied to obtain values of GT ranging from 10^4 to 10^5.

The results of these experiments indicated that turbidity removal efficiency, at any slow mixing GT level, was strongly dependent on the coagulation-flocculation temperature and somewhat less dependent on the slow mixing speed. The effect of mixing speed was more significant at the lower temperature. Optimal values of mixing speed and mixing time determined for each temperature are shown in Table E.1. Unless specified otherwise, the optimal slow mixing conditions for a given temperature were employed in all other jar tests.

Table E.1. Optimal Slow Mixing Conditions for Given Jar Test Temperatures

Temperature C°	Optimal Mixing Speed, rpm	Optimal Mixing Time, min.	Optimal GT
1	40	17.4	4.17×10^4
13	40	18.6	5.31×10^4
25	30	21.2	4.83×10^4

Initial pH. A series of experiments was conducted to determine the influence of initial pH on turbidity removal. Initial pH levels of samples were established within approximately the same range of raw pH values found in the Republic Mine tailings basin overflow over a period of one year– this range being from 7.3 to 8.1. Tests were conducted using alum dosages of 50 and 100 ppm at temperatures of 1 and 13°C. The results clearly indicated an optimal initial pH close to 7.5. Unless specified otherwise, this pH value was employed for all subsequent experiments.

Particle Electrokinetic Properties. At the outset of the study, it was intended to investigate particle electrokinetic properties using a Waters Associates Streaming Current Detector. However, Waters Associates had temporarily discontinued the manufacture of this instrument. Consequently it was not available and not employed in this study. Subsequently an attempt was made to investigate particle charge phenomena using the colloidal titration technique (27). However, this effort was unsuccessful as the necessary reagents could not be obtained from the Japanese manufacturer.

APPENDIX F
Glossary of Terms

acre ft.:	43,560 ft^3
argillaceous:	clayey
autogenous grinding:	Process in which grinding is achieved in a tumble mill through interparticle contact between the particles being ground.
ore beneficiation:	Process by which the economic value of an ore is enhanced through concentration
second-foot-day (sfd):	86,400 ft^3 = 1.983 acre. ft.
snowpack:	layer of snow covering the land surface
seepage:	Subsurface movement of water.
snowpack water equivalent:	Height of a column of water (covering a given area) with a weight equal to the weight of the snowpack covering the same area.
supra-colloidal solids:	Suspended solids consisting of particles with diameters ranging from 0.5 microns to 10 microns.

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